fording
COAL LIMITED

DEMONSTRATION AND EVALUATION OF BLAST CASTING IN A PRODUCING CANADIAN SURFACE COAL MINE

FINAL REPORT

By
FORDING COAL LIMITED

Work on this project was conducted
under the auspices of the
Department of Energy, Mines and Petroleum Resources

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EXECUTIVE SUMMARY

A contract resulting from the Request for Proposal (RFP) entitled "Demonstration and Evaluation of Blast Casting in a Producing Canadian Surface Coal Mine" was awarded to Fording Coal Limited on June 27, 1986. The project was divided into three phases under the headings: Technical Review; Testing Program; and Data Compilation, Report Preparation, and Debriefing.

The Phase 1 - Technical Review developed background information from several sources. Initially a literature search by A. Bauer (Alan Bauer and Associates Inc.) collected relevant technical papers. A site visit to two operating mines in Colorado was undertaken in the company of the Scientific Authority, Mr. H.G. Naidu. Finally, site visits from Dr. A. Bauer and Dr. T. Hagan (Golder Associates) were made. Based on the information assembled, preliminary designs for the primary blast sites were established.

The Phase 2 - Testing Program commenced in the fall of 1986. Three cases of blast casts dependant on physical geometry were evaluated during the course of the test program. These included: (A) Casting hanging wall waste off steeply dipping seams; (B) Topographic blast casting; and (C) Casting waste into mined out cuts. The final case is the most universally applicable case at Fording, where steep walls in dragline cuts provide optimum geometry. Highwall push-backs in expanding pits generate similar opportunities.

A total of nine blasts were shot during the testing phase. Operating constraints, scheduling problems, and production shutdowns resulted in considerable delays to the testing program which was not completed until December 9, 1987.

Phase 3 - Data Compilation, Report Preparation, and Debriefing followed the Testing Program. Approximately three months was required to analyze the data and produce a report.

The technical analysis commences with a review of parameters affecting blast casting. Blast cast optimization involves minimizing the sum of the cost of drilling and blasting the waste to be cast plus the cost of removal of waste volumes remaining after the shot. Blast cast variables such as explosive type, rock properties, physical blast geometry, blast hole geometry, and blast dynamics are defined as fixed or manipulative.

The net value of a blast cast is the difference between incremental costs and incremental benefits. Incremental costs are attributable to higher powder and energy factors (ie. additional drilling, accessories, and explosive costs). The incremental benefit is derived from the incremental waste volume effectively cast relative to a conventional production blast and the applicable unit cost for waste removal (by equipment). Open blast casts are defined as blasts whose cast volumes cannot be determined without applying an estimated swell factor to unsuccessfully casted waste.

Blast cast reports including introduction, general design information, pattern design information, and analysis were produced for each test blast. Blast reports as well as incremental costs and benefits are tabled in each blast analysis section. Due to confidentiality, the expressions for incremental costs and benefits contain unspecified economic parameters (ie. unit costs), however, net savings per BCM are specified.

Only two of the test blasts resulted in a calculated net savings. According to the economic models utilized, the largest savings generated was 0.106 $/BCM and the largest losses in two blasts were in excess of 0.095 $/BCM. Several blasts were marginally unfavourable. A sensitivity analysis on swell factor, percentage cast for conventional production blasts, drilling costs, explosive costs, and equipment operating costs concludes that swell factor estimation can be significant in determining net savings. Although net savings appears to be insensitive to the estimate of percentage cast for conventional production blasts, a large estimation interval suggests that it may be a relatively important variable.

There is a positive correlation between depth to width ratio (D/W) and the percent cast, although no quantifiable relationship was evident on the basis of the limited testing which introduced several variables. These variables included the cost of achieving increases in bulk strength with wet hole explosive types versus dry hole products, and blast cast effectiveness relative to toe burden on the front row.

Although outside the formal scope of the project, information relating to the drill program proved of interest. An increase in bit diameter of the three Fording Bucyrus Erie 45-R drills from 250 to 270mm was carried out during the blast cast program with an overall increase in productivity resulting. Also, following unsuccessful attempts at angle hole drilling with a contract down-the-hole rig, a Fording 45-R was converted to angle hole capability with good results.

Blast cast simulations as provided by CIL's Sabrex program can be effectively utilized to model potential applications.

The blast cast project has successfully illustrated a method for assessing blast casts. Guidelines regarding sensitive blast parameters have been developed. Although the problem of optimization has not been fully addressed, the economic potential for blast casting in Canadian surface coal mines has been demonstrated.

Le 27 Juin 1986, la société Fording Coal Limited a reçu un contrat comme conséquence d'une demande de proposition intitulée "Démonstration et évaluation du sautage de projection ("blast casting") dans une mine canadienne à ciel ouvert en production".

Le project comporte trois phases:

1. Révision technique

2. Programme d'expérimentation (d'épreuves)

3. Compilation des données et comptes rendu des resultats.

Phase 1:

Il s'agit de faire une révision approfondie des techniques et des informations recueillies de plusiers sources. A. Bauer (Alan Bauer and Associates Inc.) a complété la recherche de documents et techniques pertinents.

Une visite sur les lieux à deux mines en exploitation au Colorado a été entreprise en compagnie de l'autorité scientifique M.H.G. Naidu. Par la suite les Drs. A. Bauer et T. Hagan (Golder Associates) ont fait de même. Des projects preliminaires basés sur les renseignements ainsi obtenus ont été formulés pour les sites de sautages primaires.

Phase 2:

C'est en automne 1986 que commencera le programme d'expérimentation (d'épreuves). Trois exemples de sautages de projection en fonction de la géométrie physique ont été évalués au cours du programme préliminaire. Ceux-ci comprennent:

A. Le déblayage du "Casting Hanging Wall" des couches à pente raide;

B. "Topographic Blast Casting" et;

C. Le déplacement (le déblayage) des stériles dans les lieux déjà exploités.

C'est ce dernier cas qui s'applique le plus particulièrement chez Fording où des murs à pente raide résultent des coupes de "dragline" et offre une géométrie optimum. "High wall push-backs" dans des mines en expansion offrent des avantages semblables.

Pendant la phase d'expérimentation un total de neuf sautages ont été faits. Le programme n'a été complété que le 9 décembre 1987, à cause des problèmes résultant des contraintes imposées par la production, des difficultés d'horaire et des arrêts de travail.

Phase 3:

La compilation des données et la rédaction du compte rendu des résultats ont fait suite au programme d'épreuves.

Il a fallu près de trois mois pour completer l'analyse des donnèes et pour rédiger le rapport.

L'analyze technique débute par une révision des paramètres influençant le sautage de projection. Pour obtenir les optima de rendement de sautage de projection il est necéssaire de réduire au minimum les coûts de forage et de sautage en plus de ceux de déblayage des stériles après la mise à feu. Les coûts variables de sautage de projection résultent du type d'explosif utilisé, les propriétés du roc, de la géométrie du sautage, de la géométrie du trou de sautage et de la dynamique du sautage. Ces variables peuvent être définis soit comme fixes soit sujets à la manipulation.

La valeur nette d'un "blast cast" est la différence entre l'augmentation des coûts et celle des bénéfices. L'augmentation des coûts d'exploitation est généralement attribuable au coût de l'explosif et de l'énergie requise (tels que le forage additionel, les accessoires et les explosifs). Un bénéfice accru résulte du volume de stériles déplacés en comparaison avec celui d'un sautage ordinaire et le côut par unité pour le déblaiement des stériles (par l'utilisation d'équipment). Les sautages de projections ouverts sont définis comme des sautages dont le volume de stériles ne se determinent pas sans l'application d'un estimé dit facteurs de gonflement comparé aux stériles non déblayés.

Le compte-rendu de sautage de projection comporte l'introduction des renseignements du design genéral, des détails sur le design du patron et une analyse des résultats des sautages. En plus il contient un résumé de l'augmentation des coûts ainsi que des bénéfices.

Du au caractère confidentiel des recherches, l'augmentation des coûts et des benéfices est exprimée en paramètres économiques non détaillés (C.A.D. coût par unité). En même temps les bénéfices nets par mètre cube en place (BCM) y sont spécifiés.

Seulement deux des sautages ont réussi à nettement produire des économies. Selon le modèle économique choisi, les économies les plus marquées étaient de 0.106 $/BCM et les pertes les plus considérables en deux sautages dépassaient le 0.095 $/BCM. Plusiers sautages étaient légèrement défavorables. Une analyse de sensitivité sur le facteur de gonflement, le pourcentage de projection pour un sautage ordinaire de production, les frais de forage, le coût des explosifs, et les coûts pour le fonctionnement de l'équipment porte à conclure que l'estimé du facteur de gonflement peut avoir un impact significatif pour déterminer les économies obtenues.

Quoique les économies nettes paraissent ne pas avoir de rapport à l'estimé du pourcentage de déblayage pour le sautage conventionnel de production, un grand écart d'estimation suggère que cela pourrait être une variable d'importance.

Il y a une correlation positive entre le rapport profondeur/largeur (depth/width ratio) et le pourcentage de projection, même si aucun rapport quantifiable n'était évident sur la base d'épreuves limitées qui presentent plusieurs éléments variables. Les variables incluent le coût d'accroître la force avec des explosifs pour tous contenant de l'eau vs des produits pour les trous secs, et l'efficacité du sautage de projection par rapport à la charge de fonds sur la première rangée.

Quoique ce soit en dehors de la portée du projet, des statistiques se rapportant au programme de forage ont suscité de l'intérêt. Une augmentation du diamètre de la mèche des trois foreuses Bucyrus-Erie 45R de Fording, de 250 à 270 mm pendant le programme du sautage de projection a abouti à une augmentation de production. Aussi, suivant des tentatives sans succes de forer des trous inclinés avec une foreuse de contrat de type "down-the-hole", une 45R de Fording a été modifiée pour obtenir de tels trous avec de bons resultats.

Des sautages de projection simulés tels que prévus par le programme SABREX de CIL peuvent être utilisés avantageusement pour servir de modèle pour les applications ultérieures.

Le projet a réussi à démontrer une méthode pour estimer les coûts des sautages de projection. Des lignes de conduite concernant les paramètres qui affectent le plus les sautages ont été élaborées. Bien que le problème d'optimisation n'ait pas été complètement établi, le potentiel économique pour le sautage de projection au Canada dans les mines de charbon à ciel ouvert a été démontré.

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DEFINITIONS

OB - Overburden material - natural soils or rehandle.

LSM - Loose Square Meters

BSM - Bank Square Meters

CM - Cubic Meters

LCM - Loose Cubic Meter

BCM - Bank Cubic Meter

S - Swell Factor

LRB - Last Row Blasted

BI - Blast influence

DTH - Down the Hole Delays (Nonel, Period 10 = 340 ms)

PF - Powder Factor (kg/BCM)

EF - Energy Factor (weu/BCM)

VOD - Velocity of Detonation

V_T - Total Volume of Cast Blast

V_C - Volume effectively cast (ie eliminating equipment rehandling)

%Vc - Fraction cast relative to total volume = Vc/Vt

%Vpc - Estimated percent cast in a standard production shot/100

Vpc - Estimated Production Cast Volume (ie volume cast without any incremental costs - standard production drilling & loading) = Vt * %Vpc

U - General symbol for unit cost

U_{TS} - Unit cost of casting waste ($/BCM) when the loading equipment is a shovel side casting the waste.

U_{DL} - Unit cost of casting waste ($/BCM) when the loading equipment is the dragline.

U_{DLM} - Unit benefit of casting waste beyond M seam but infront of L seam (Dragline Mining)

U_{DLL} - Unit benefit of casting waste beyond L seam (Dragline Mining)

U_D - Unit cost of drilling 1 meter of hole including redrills
over drilling.

U_N - Unit cost of down-the-hole delays (Nonels)

U_P - Unit cost of primers

U_{ANFO} - Unit cost of 1 kg. of ANFO

$U_{5\%}$ - Unit cost of 1 kg. of 5% ANFO

$U_{10\%}$ - Unit cost of 1 kg. of 10% ANFO

U_{PGA} - Unit cost of 1 kg. of powergel A

U_{PGB} - Unit cost of 1 kg. of powergel B

U_{2495} - Unit cost of 1 kg. of SuperAN 2495

U_{2505} - Unit cost of 1 kg. of SuperAN 2505

U_{2510} - Unit cost of 1 kg. of SuperAN 2510

U_{2515} - Unit cost of 1 kg. of SuperAN 2515

U_{MAG} - Unit cost of 1 kg. of Magnafrac 9000

U_{9013} - Unit cost of 1 kg. of Magnafrac 9013

U_{9020} - Unit cost of 1 kg. of Magnafrac 9020

1.0 INTRODUCTION

1.1 REQUEST FOR PROPOSAL

A Request for Proposal (RFP) dated April 9, 1986 was issued by the Department of Supply and Services, Canada on behalf of the Department of Energy, Mines and Resources, Centre for Mineral and Energy Technology (CANMET). The RFP (DSS File No. 09SQ.23440-6-9022) was entitled "Demonstration and Evaluation of Blast Casting in a Producing Canadian Surface Coal Mine (a cost shared project).

1.2 PROPOSAL

Following a review of the RFP, Fording Coal Limited submitted a proposal to Supply and Services, Canada dated May 7, 1986. As outlined in the RFP, the proposal included a Technical Proposal, Price Proposal, Management Proposal, and Method of Payment.

It was proposed that all test work and engineering would be performed at Fording Coal Limited's Fording River Operation located at Elkford, B.C. The mine was ideally suited for the proposed test program in that both truck/shovel and dragline methods were utilized to recover coal from variable dipping seams. It was anticipated that test sites could be selected to demonstrate the applicability of blast casting to prairie as well as mountainous coal operations.

The proposal was submitted on a firm price basis of $94,102.21. The costs of the program were expected to include:

1. Labour
2. Material and Supplies
3. Equipment Rental
4. Subcontractors (Review Consultants)
5. Travel and Living Expenses
6. Computing Costs

7. Typing
8. Drafting
9. Communications

1.3 CONTRACT AWARD

The contract was awarded on June 27, 1986 by the Department of Supply and Services. The original schedule called for finalizing of the program in July 1986 with a six month field trial from August, 1986 to January, 1987. The submission of the final report was scheduled for March 31, 1987. Delays, which are detailed in Section 1.6.2 led to a contract extension to February 19, 1988.

1.4 FORDING COAL LIMITED/FORDING RIVER OPERATIONS

Fording Coal Limited is one of Canada's largest producers of metallurgical coal. With head office in Calgary, and operations in Alberta and British Columbia, the Company is a wholly owned subsidiary of CP Limited.

The Fording River Operation is located 29 miles northeast of Elkford, in the Fording River Valley of British Columbia (Figure 1). The mine's reserve consists of over 300 million tonnes clean coal in 15 seams ranging from 1 to 10 meters thickness. Production commenced in 1972 at 3 million tonnes per year. Current capacity is 50 million bank cubic meters (BCM) waste producing in excess of 5 million tonnes clean coal to a varied domestic and foreign market. The coals range in rank from medium to high volatile bituminous and are utilized as both metallurgical and thermal products.

1.4.1 Geology

The general geology of the area consists of two north-south trending synclines in the non marine Kootenay Group, with one on each side of the Fording River. These synclines are separated by the Ericson fault, a normal fault with a vertical displacement of 300 meters up to the east.

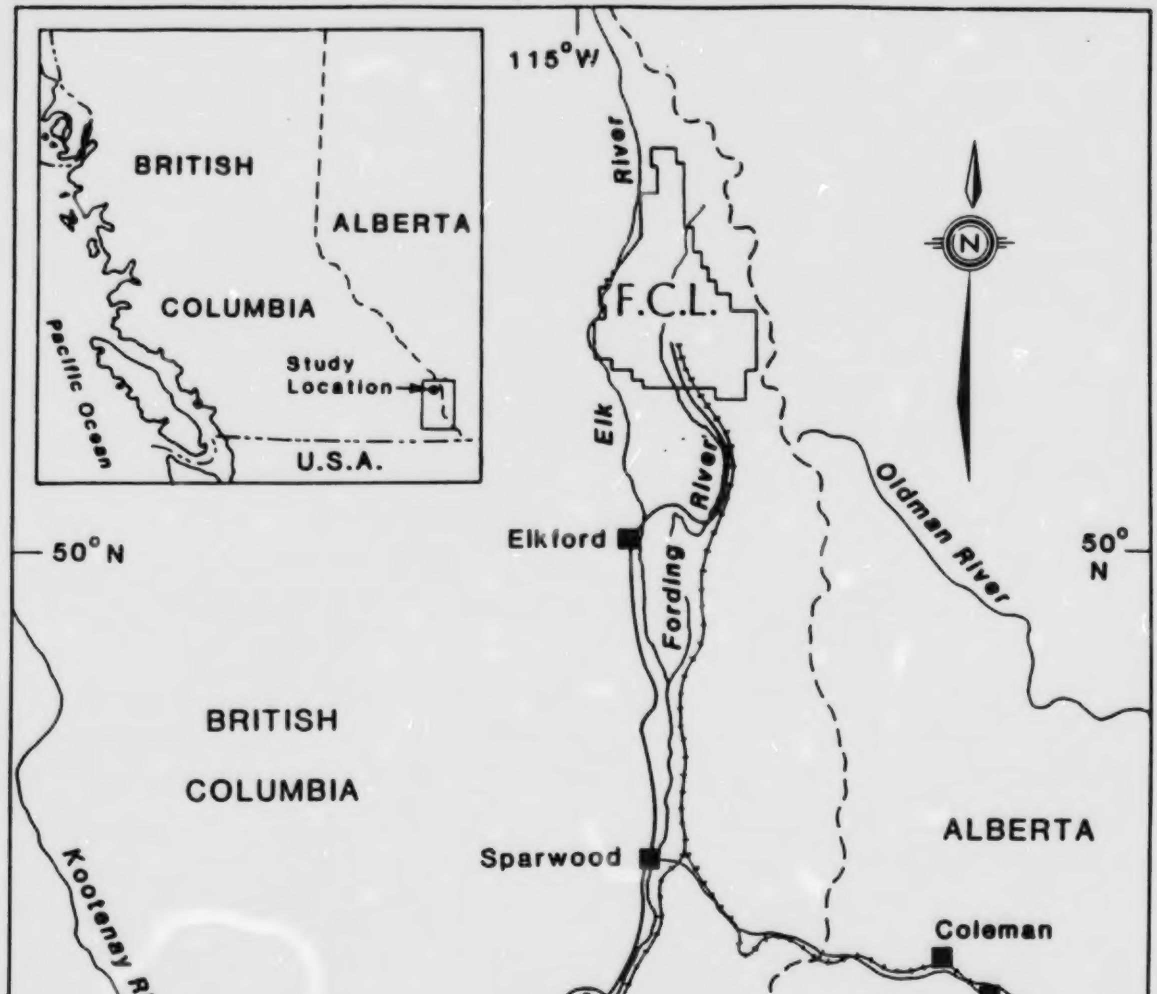

BRITISH
COLUMBIA
ALBERTA
Pacific Ocean
Study Location
U.S.A.
115°W
Elk River
F.C.L.
Fording River
Oldman River
Elkford
50° N
50° N
BRITISH
COLUMBIA
ALBERTA
Sparwood
Coleman
Blairmore
C.P.R.
Kootenay River
3
C.P.R.
Fernie
Cranbrook
F.C.L. - Fording Coal Ltd.
coal holdings
93
115°W
Scale (kilometers)
0
12.5
25
50

Figure 1

Location Map

The Fording River Valley is a result of the erosion of the intervening anticline along the locus of the Ericson fault (Figure 2).

The majority of rock in the mining areas is comprised of a 550m thick sub unit of the Kootenay Group known as the Mist Mountain Formation. Rock within this formation range from mudstones with uniaxial compressive strengths of 25 MPa to coarse grained sandstones with compressive strengths as high as 200 MPa. Joint sets are approximately orthogonal to bedding with spacing ranging from 10 cm to several meters. Bedding thickness is extremely variable.

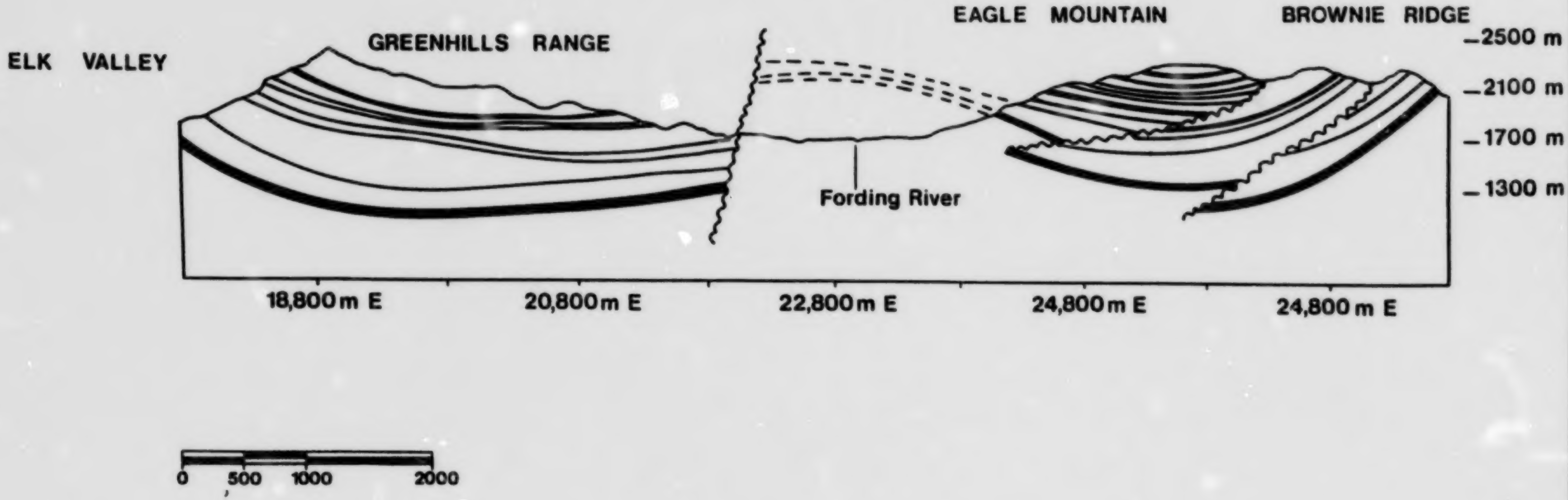
1.4.2 Mining

Production is achieved with a combined truck/shovel and dragline operation from several separate mining areas. Generally, development and benching is carried out with five 23 cubic meter and three 11.5 cubic meter shovels combined with fifty-nine 109 and 154 tonne haul trucks. Topography varies from valley bottom to contour (hillside), to ridge/mountain top. The truck/shovel fleets are presently being consolidated in a large area known as Eagle Mountain.

Lower elevation subcrops and pit bottoms are frequently mined with a 46 cubic meter dragline supplementing truck/shovel production. Extended benching in a series of parallel cuts is commonly practiced, with seam dip varying from 0 to 25°.

Fording's current waste productivity per manshift ranks among the highest in the world. Recently Fording River Operations was awarded the prestigious silver medal at the Canadian Business Awards. This high productivity is a result of many factors including development programs in critical areas such as drilling and blasting. To this end, the "Demonstration and Evaluation of Blast Casting in a Producing Canadian Surface Coal Mine" (Blast Cast Project) is a practical means of refining knowledge in this area with the objective of further improving productivity and moving material at the lowest overall unit cost.

Figure 2 General Geological Section


ELK VALLEY
GREENHILLS RANGE
EAGLE MOUNTAIN
BROWNIE RIDGE
-2500 m
-2100 m
-1700 m
-1300 m
Fording River
18,800 m E
20,800 m E
22,800 m E
24,800 m E
24,800 m E
0
500
1000
2000

1.5 PROJECT TEAM

The project team is comprised of an in-house group supplemented by outside consultants. Several changes in the team have occurred over the life of the project, for various reasons. A summary of the original project team and replacements is shown in Figure 3. The replacement of personnel as outlined in a letter dated October 22, 1986 was reviewed by the Scientific Authority and found acceptable.

Figure 3 - PROJECT TEAM

TITLE	NAME	COMMENTS	REPLACEMENT
Project Manager	D.A. Kennedy		
Project Engineer	D.A. Brown		
Blasting Engineers	G. Clausen	Transfer/Termination	R. Gold
	J.A. Currie	Termination	R. Gold
Mining Engineer	G. Trout	Addition to project team	
Technicians	R. Krishan		
	P. Daum	Termination	D. Swail G. Tarrant
Consultants	Dr. T. Hagan	Availability limited (overseas)	
	Dr. A. Bauer	Deceased	

1.6 PROJECT OVERVIEW

The Blast Cast Project was divided into three phases under the following headings: Technical Review; Testing Program; and Data Compilation, Report Preparation and Debriefing.

1.6.1 Phase 1 - Technical Review

A technical review was undertaken to develop background information on the subject of blast casting. Initially, a literature search was carried out by A. Bauer (Alan Bauer & Associates Inc.) to collect relevant technical papers and to review design parameters, performance criteria, and experiences at other operations (primarily in the U.S.). This was completed in August 1986.

A review of test sites was also carried out. The site analysis involved geological interpretation and cross section generation, and discussions regarding operating, scheduling, and technical concerns. It was concluded that the majority of sites originally selected with the exception of 3 Pit were suitable for testing. The areas in 3 Pit which had been regarded as suitable test sites, were no longer available by the time testing commenced.

The primary sites were to include Eagle Mountain, Brownie Pit, and Lake Pit. Site evaluation was conducted and approved by the Scientific Authority. A property map showing the active test sites is represented in Figure 4.

A site visit to two operating mines practicing blast casting was undertaken by D. Kennedy and G. Clausen in the company of the Scientific Authority H.G. Naidu in August 1986. Drilling and blasting practices relevant to blast casting at the Colowyo Coal Co., Meeker, Colorado, and the Trapper Mining Inc., Craig, Colorado were reviewed with operating personnel. Mining equipment and methods were also discussed. It was concluded that both operations realize substantial savings by casting between 35 - 45% of the dragline cut material. Angle drilling played a large role in effective casting. Both operations utilized Bucyrus Erie 45-R drills with 270 mm (10 5/8 inch) bits. The trip reports are included in Appendix A.

The technical review was further strengthened by site visits from Dr. A. Bauer (Alan Bauer & Associates Inc.) and Dr. T. Hagan (Golder Associates, Australia) who were requested to review Fording's proposed test sites and to make recommendations regarding blast casts at these sites. These individuals' practical experience combined with information from the literature search and the Colorado field trip provided Fording personnel with a starting point for the field test program.

The Phase 1 - Technical Review was originally scheduled for completion in August 1986. Late receipt of the consultant's reports contributed to a minor delay.

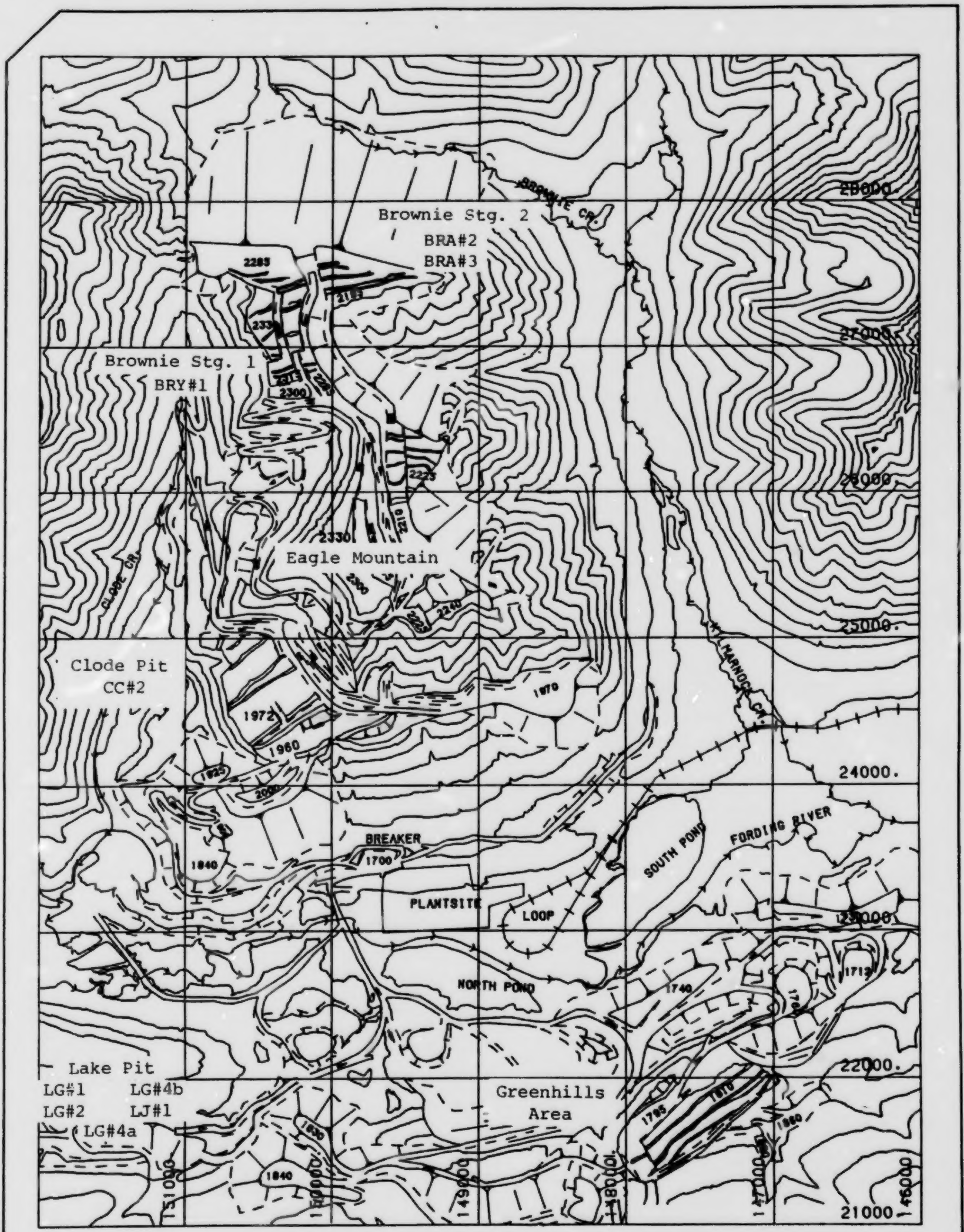

BROWNIE CR.
Brownie Stg. 2
BRA#2
BRA#3
2283
2193
2330
Brownie Stg. 1
BRY#1
2315
2300
2225
2210
2330
Eagle Mountain
CLODE CR.
2300
2240
2225
Clode Pit
CC#2
1970
1972
1960
1925
2000
KILMARNOCK CR.
BREAKER
1700
1840
SOUTH POND
FORDING RIVER
PLANTSITE
LOOP
NORTH POND
1740
1713
1700
Lake Pit
LG#1
LG#2
LG#4a
LG#4b
LJ#1
Greenhills
Area
1800
1840
1795
1810
1860
28000.
27000.
26000.
25000.
24000.
23000.
22000.
21000.
151000.
150000.
149000.
148000.
147000.
146000.
SCALE 1 : 35,000
0
0.5
1.5
2.5
(kilometers)
Z
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Figure 4

Property Map

1.6.2 Phase 2 - Testing Program

It was originally anticipated that Eagle Mountain would be a primary test site for topographic (steep terrain) blast casts. A blast cast was designed and field control implemented for the 2330 Bench of Eagle Stage 4 in August 1986. A rental drill unit was contracted on an hourly basis to drill angle holes for the test program. The drill was a truck mounted Schramm equipped with a down-the-hole hammer and 254 mm (10 inch) bit. The contractor was unable to drill angle holes effectively on this initial pattern and the site was abandoned. This was unfortunate since this site represented one of the better areas to test casting on steep terrain. Coal markets and operating constraints further affected the availability of this area: in the fall of 1986 three 23 cubic meter shovels operated on the bench simultaneously, forcing production drilling to be priorized. Subsequent development of the 2315 Bench of Eagle Stage 4 was deferred until 1988. As a result, topographic (steep terrain) tests were shifted to Brownie Pit Stage 2.

Although 3 Pit had initially been considered as an area to evaluate "up-dip" casting of waste, the opportunity was lost be the time the field testing program commenced. Similar "up-dip" casting opportunities existed in Brownie Pit Stage 2, although this area did not include blasting to the top of coal. Further, "up-dip" casting potential was later utilized at Clode 9 Seam Pit.

Test work proceeded as scheduled in Brownie Pit Stage 1 in the fall of 1986. The first blast, BR-Y #1 was intended to cast hanging wall waste from steeply dipping 13U Seam. Drilling was carried out by contract drill for both angle and vertical holes. Although the productivity in angle hole drilling was low, the pattern was successfully completed and blasted on October 1, 1986.

The 2195 Bench in Brownie Pit Stage 2 was the site for two blasts with the objective of casting waste from a mountain ridge into Brownie Creek valley. BR-A #2 was drilled by contractor with a combination of angle and vertical holes and was initiated on October 29, 1986. BR-A #3 was drilled by contractor with exclusively vertical holes and was blasted on November 14, 1987.

The lack of test sites due to operating constraints, scheduling delays, and production shutdowns resulted in a long hiatus in the testing program. A delay in the Marion 8400 dragline move from 3 Pit to Lake Pit in particular, limited test sites through the winter, spring, and summer of 1987. Production shutdowns due to market conditions also delayed the continuation of field testing. The production shutdowns, totalling 68 days, are listed below:

December 19, 1986 - January 16, 1987
March 13 - 20, 1987
June 27 - July 27, 1987

The dragline move from 3 Pit to Lake Pit in September 1987 afforded substantial opportunities to cast waste from Cut #2 into the mined out Cut #1. This form of blast casting in dragline cuts is regarded as having the best long term potential at Fording River Operations and is undoubtedly the most universally applicable; thus its role in the test program warranted emphasis. A total of four test blasts in Lake Pit Cut #2 (3 independent shots) were initiated between September 18, 1987 and October 17, 1987.

During November 1987, a Fording Bucyrus Erie 45-R drill was converted to angle hole capability in order to test the effectiveness of angle hole drilling with a rotary rig and examine the potential of angle holes relative to vertical hole blast casting. This change-over was initiated following a review of the contract drilling the previous fall. The decision to convert a 45-R drill to angle holes was made in July 1987. Although a purchase order for parts was issued in the same month, the materials were not received until early November. This represented a substantial unanticipated delay. The change-over time was approximately one week.

A combination of angle and vertical holes were applied to Clode 9 Seam Pit C-C #2 blast which was initiated on November 26, 1987. A final test blast L-J #1, again utilizing angle and vertical holes, was drilled in Lake Pit Cut #3 and was fired on December 9, 1987. In both cases, waste was cast into a previously mined area. The results of the angle hole rotary drilling were relatively good, and are elaborated on in Section 2.3.

A summary of the test program is shown in Figure 5.

The testing program was originally scheduled for completion in February 1987. Based on the loss of test sites in Eagle Mountain, an extension to May 1987 was made in October, 1986. A further extension to October 1987 was required following a review of production shutdowns and the availability of test sites. The delayed dragline move to Lake Pit in particular affected the completion of the testing program. A final extension to the test program was necessitated on the basis of the angle drill conversion, and the desire to include these results within the project scope. The testing program was completed when the last blast (L-J #1) was fired on December 9, 1987.

1.6.3 Phase 3 - Data Compilation, Report Preparation and Debriefing

Data was recorded and analyzed on an ongoing basis throughout the testing program. The test results included still photographs and survey pickups of blasts, as well as VCR and high speed photography (Locam) of several shots. A 35mm SLR camera was used to document the overall physical layout of the pattern areas, before and after the blasts. A selection of the resulting photographs is included in Section 2.3.

The survey work involved establishing the relative position of all blast holes on the pattern as well as determining the before and after physical topography. Section lines were run in the field following blasts to establish the muck profile created by the cast. This information was used extensively to analyze the effectiveness of the cast blast and to generate

Figure 5: SUMMARY OF BLASTS

BLAST	PIT/LOCATION	DATE	OBJECTIVE	DRILLING	DOCUMENT-ATION
BR-Y #1	Brownie Stage 1 300 Bench	1 Oct./86	Cast hw waste from steeply dipping 13u seam	WHA 10" dia. holes. Crest row alternating vertical with 10' extra subgrade and 10° angle.	
BR-A #2	Brownie Stage 2 2195 Bench	29 Oct./86	Cast waste from mountain ridge (topography) to valley.	WHA 10" dia. holes. Crest row @ 10° with 10' extra subgrade. Second row @ 10°. Others vertical.	VHS (failed)
BR-A #3	Brownie Stage 2 2195 Bench	14 Nov./86	Cast waste from mountain ridge (topography) to valley.	WHA 10" dia. holes. Vertical holes - 2 rows only.	
L-G #1	Lake Pit #2 Cut	18 Sept./87	Cast hw 'M' seam into previously mined dragline cut	45-R 10 5/8" dia. holes Vertical Sept. 8 - 17	VHS
L-G #2	Lake Pit #2 Cut	24 Sept./87	Cast hw 'M' seam into previously mined dragline cut	45-R 10 5/8" dia. holes Vertical Sept. 17 - 24	Locam
L-G #4a	Lake Pit #2 Cut	17 Oct./87	Cast hw 'M' seam into previously mined dragline cut	45-R 10 5/8" dia. holes Vertical Oct. 4 - 13	Locam
L-G #4b	Lake Pit #2 Cut	17 Oct./87	Cast hw 'M' seam into previously mined dragline cut	45-R 10 5/8" dia. holes Vertical Oct. 4 - 13	Locam
C-C #2	Clode 9 Seam 1805 Bench	26 Nov./87	Cast waste into mined out cut	45-R 10 5/8" dia. holes 20° and 10° angle and vertical holes. Nov. 13 - 25	Locam & VHS
L-J #1	Lake Pit #3 Cut	9 Dec./87	Cast hw 'M' seam into previously mined dragline cut	45-R 10 5/8" dia. holes 20°, 15°, 10° angle and vertical holes. Nov. 18 - Dec. 1	Locam & VHS

data for economic analysis (ie. volumes, profile, and mining situation). A total of 25 survey man hours was utilized above and beyond standard procedural pickups.

The VCR and Locam photography was in some cases limited by the terrain and vantage point. No record of the BR-Y #1 and BR-A #3 blasts was possible due to their position. An equipment malfunction also affected the VCR coverage of BR-A #2. Blast L-G #2 was filmed successfully with a VCR. The final 2 blasts C-C #2 and L-J #1 were also filmed with a VCR.

The availability of a high speed Locam camera created some concern early in the project. Although a unit was on site prior to the L-G #1 Blast in September 1987, it was required elsewhere at the time of initiation. The final blasts following L-G #1 were filmed with Locam units from either McGill University or CIL Inc. Technical personnel from CIL assisted in the Locam filming of all blasts. Their research facility in Toronto also provided analysis. The best Locam results were obtained with the final two shots C-C #2 and L-J #1 where face velocities were calculable. Delays of 4 - 6 weeks in the developing and analysis of each blast film precluded their use as an analytical tool in the optimization of subsequent blasts.

A summary of the VCR and Locam documentation is included in Figure 4. Highlights of the films have also been edited onto a single VHS tape which is included with the report.

The majority of compilation could not commence until late in the project when adequate comparative test results were available. The bulk of this work took place in late November, December, and January. Report preparation was carried out concurrently in January, and completed in February 1988. The draft final report was made available to the Scientific Authority by February 29, with the final report to follow upon approval of the draft. The debriefing seminar was scheduled for mid March at the time of writing.

The delivery of the final report was delayed in part to problems associated with the field testing program. Delays within Phase 3 were mainly attributable to the fact that the majority of testing took place late in the program requiring a great deal of analysis and compilation with fixed resources. Manpower reductions through attrition as noted in Section 1.5 also contributed to the delay. Although the majority of work was handled by a single blasting engineer it is interesting to note that Phase 3 required approximately double the manhours originally scheduled. Total time for Phase 3 - Data Compilation, Report Preparation and Debriefing was approximately 3 months not including the time required to set up the debriefing session. This was longer than the 2 months originally scheduled.

2.0 TECHNICAL ANALYSIS

2.1 GENERAL

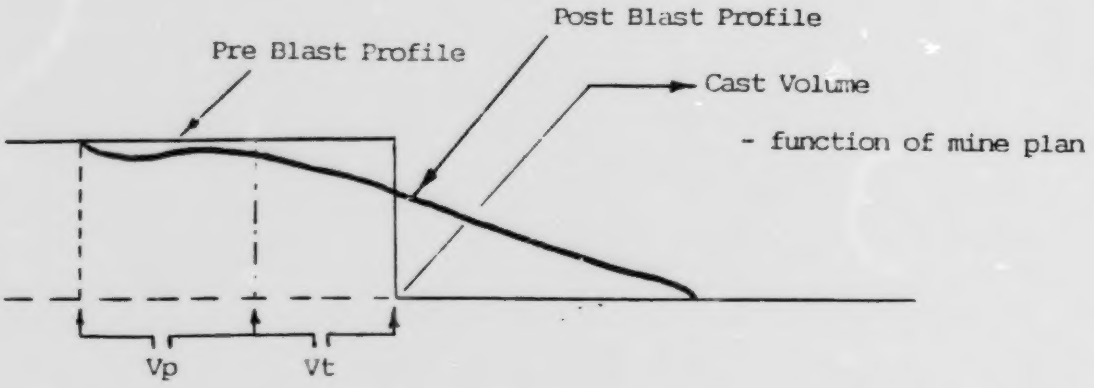
A discussion regarding blast casting relative to overall blast optimization and mining economics is required in order to establish a consistent means of analysis. This leads naturally to a review of the fundamentals of blast casting (ie blast variables). The applicability of blast casting to various physical mining situations at Fording River Operations is briefly covered.

2.1.1 Blast Casting: Part of the Overall Blasting Optimization

Total blasting optimization can be reduced to the process of minimizing total mining costs. Total mining costs can be adequately expressed as the sum of drilling and blasting costs plus the cost of removing the waste rock. Figure 6 illustrates in a conceptual way the total economics of drilling, blasting, and waste removal. The total minimization problem has two components. One component is associated entirely with production blasting and the other with blast casting. The test work and subsequent analyses are directed exclusively at the blast casting process.

Simply stated blast cast optimization involves minimizing the sum of the cost of drilling and blasting the waste volume to be cast plus the cost of removal of waste volumes remaining after the shot. The complex nature of the problem is evident. Total costs decrease as the cast volume (Vc) increase. However in order to increase the cast volume the cost of drilling and blasting often increases. The other factor that affects the total mining cost is the cost of removing unsuccessfully cast waste. This unit cost is a function of operating cost and equipment productivities. Cast blasting potentially lowers operating cost and raises equipment productivities if significantly higher energy factors (EF) are used. Operating costs drop in response to reduced maintenance costs; productivity increases as a result of improved digging conditions.

It is difficult to quantitatively assess the change in the unit cost of waste removal as a function of the unit drilling and blasting cost. It is equally difficult to understand the relationship between drilling and blasting costs and the volume cast. Several blast variables affect each of these relationships. Many of these variables are manipulable and many are fixed in a given situation. Once clear relationships are defined between the various blast parameters which influence blast casting, the task of minimizing costs is reduced to selecting values for appropriate manipulable variables for a given set of fixed variables in order to reduce costs. The key is to recognize a situation (a set of fixed conditions) which are suitable for blast casting benefits. In many instances the potential for reducing total costs by increasing D&B costs result in only minimal cast volumes and zero or negative benefits. In many instances significant waste volumes may be cast without increasing D&B costs at all. This may or may not be the optimal design. Although volumes of material cast in a standard production shot have no additional costs, cast volumes may be so small that the total benefit is small. Alternatively, with moderate increases in D&B costs, larger volumes could be cast resulting in a lower overall cost even though the savings for each bank cubic meter (BCM) cast is reduced.



The above schematic represents a cross section through a blast pattern where the front portion with volume, Vt, has been drilled and loaded for a cast blast. The pattern area behind this is a standard pattern with volume, Vp.

Let:

Upr = Unit cost per BCM of drilling and blasting - production pattern
Uc = Unit cost per BCM of drilling and blasting - Blast cast pattern
Ue = $\frac{\text{opcost/hr.}}{\text{productivity/hr.}}$ = Equipment operating cost per BCM
Uep = Equipment operating cost per BCM - production blast
Uec = Equipment operating cost per BCM - blast cast
Vc = % Vc * Vt = volume effectively cast
TC = total cost

Then:

TC = cost of drilling and blasting + cost of removing waste
TC = Vp Upr + Vt Uc -Total D&B cost
+ Vp Uep -Cost of waste removal in production shot
+ (1 - % Vc) * Uec Vt -Cost of removing unsuccessfully cast waste
TC = Vp(Upr + Uep) + Vt[(1 - % Vc)Uec + Uc] } Eqn. 1

Production Blasting Analysis **Blast Casting Analysis**

The above expression represents the total cost of production blasting and blast casting independently. This study is directed solely at minimizing the second term associated with blast casting.

Figure 6 The Complete Blasting Analysis: Economics

True optimization is beyond the scope of this test work. This test work is aimed at isolating the important blast casting variables and their relative importance. The starting point for design was derived from the literature and consultants suggestions until actual test results could be obtained. Rather than addressing the optimization problem, the easier problem of blast comparison as outlined in Figure 7 is attempted where a test blast cast is compared to a standard production blast. This problem reduces to the evaluation of incremental benefits and incremental costs due to drilling and blasting. Incremental drilling and blasting costs are easily deduced with good accuracy. Total incremental volumes cast are not easily attainable. Volumes cast from a blast cast are easily attained. Volumes cast under identical geometrical and geological situations with production drilling and blasting patterns can only be estimated. This estimation is necessary in order to make a valid incremental cost and incremental benefit analysis. The economics of a specific blast cast can be sensitive to the estimated volume cast for the reference, production blast. Sensitivity analysis will be useful in order to explore sensitivity to this estimation in each situation.

Consider 2 shots to be compared.

1) a standard production shot with volume Vp

2) a cast blast shot with volume Vt

assume that both volumes are equal so that $Vt = Vp = V$.

Evaluating the total cost of each shot from eqn. 1 (same notation used in Figure 6).

$$(TC)pr = V[(1 - \%Vpc)Uep + Upr]$$

$$(TC)c = V[(1 - \%Vc)Uec + Uc]$$

The net savings (NS) of the blast cast shot over the production shot $= (TC)pr - (TC)c$

$$NS = V[Uep - Uec - \%Vpc * Uep + \%Vc * Uec + Upr - Uc]$$

Assuming the operating cost in the cast blast is less than the operating cost in the production blast such that:

$$Uec = R * Uep \quad (0 < R < 1)$$

then $NS = V[Uep(1 - R - \%Vpc + \%Vc * R) + Upr - Uc]$

if $R=1$, the benefit is underestimated and the formulae is simplified to:

$$NS = V[Uep(\%Vc - \%Vpc) - (Uc - Upr)] \qquad \text{eqn. 2.}$$

Equation 2 states that the net savings is equal to the incremental volume cast times the unit operating cost minus the incremental cost of drilling and blasting. Note this simplified formulae underestimates the net savings.

Figure 7 Economic Comparison of a Blast Cast and a Production Blast.

The estimated values of volumes cast for the production blast can be estimated from field experience or alternatively computer simulation techniques can be used for estimation purposes.

Many of the blast casts in the test program were tied in with production blasts. From an analysis viewpoint this adds to the complexity. Ordinarily if a blast cast is initiated independently it is certain that all volumes cast originated from the blast cast. This statement may not be true if the blast cast is a subset of a production shot. As a result, blast cast volumes expressed as a percentage of the volume drilled and loaded for a shot can be very large and conceivably exceed 100%.

There are deemed three types of physical situations in which blast casting has potential at Fording River Operations. All three were attempted within the scope of the project.

A. Casting Hangingwall Waste Off Steeply Dipping Coal Seams.

By casting waste off of steeply dipping seams there is potential to save dozer costs and realize benefits of reduced coal loss and dilution.

B. Topography Blast Casting

Consider a general situation where waste can be cast off of the mining bench into a valley. Quantities of waste cast beyond the lowest design bench level would not have to be subsequently rehandled. Waste volumes cast off the existing bench but remaining as a veneer of rehandle to be mined on subsequent benches has the benefit of deferring mining costs.

C. Casting Waste into Mined Out Cuts

This situation occurs most frequently in dragline cuts at Fording River Operation. Equal potential exists for this application in truck/shovel areas when highwalls are pushed back. Benefits include reducing waste volumes to be handled and rehandled with equipment.

For a given blast cast there is a pre blast profile and a post blast profile. There is a quantifiable amount of explosives and drilling that is required to achieve the post blast profile. This information is fundamental to the analysis of each blast cast. The treatment of this information after the data has been collected is site specific.

Drilling costs can differ from pit to pit, from month to month, within a mine. Explosive types (dry or wet hole types) can change from pit to pit and within different areas of pits. With a defined physical situation, the economics of casting will be different if the area is predominantly dry compared to an area that is predominantly wet.

Given a post blast surface, the volume of material cast and therefore the percentage cast must be defined consistently with the mine plan so that valid economics can be applied. The volume cast given a specific post blast cross section may be different for a truck shovel operation or for a dragline operation. Furthermore, once the percentage cast has been defined, the unit benefit to each BCM cast will certainly depend on the mine plan as well. The benefit of eliminating one BCM from a dragline operation is different than the benefit of eliminating one BCM of waste from a truck/shovel operation.

Additionally, the physical situation immediately outside the vicinity of a blast cast will also have an impact on the feasibility of blast casting. For example, for identical hole depths, rock types and burdens, the volume cast will certainly be different if material is cast to a pit floor which extends horizontally from the bottom of the loaded holes or if the floor catching all the casted waste is 3 benches below the elevation of the bottom of the bore hole. In this case, the physical situation outside the vicinity of the blast cast has materially altered the mean flight time of casted waste. In many respects the most important summary of a blast cast is a statement of drilled meters, kilograms of various explosives and a before and after shot profile. The many site specific variables can then be factored into the analysis in order to arrive at an economic assessment. Information in this format is most useful to other operations.

2.1.2 Fundamentals of Blast Casting

There are several variables influencing the outcome of a blast cast. These variables can be divided into controllable and fixed variables. Controllable variables are variables which can be easily manipulated by the designer. Fixed variables cannot be manipulated. Figure 8 shows a list of variables along with an indication of the nature of the variable.

A general discussion on each of these variables will assist in illuminating the difficulties inherent in blast casting design. This information was extracted from the sources referenced as well as experience attained at Fording River Operations.

A. Explosive Type

Heave energy and strain energy are the two components of total explosive energy. For blast casting it is important to maximize the amount of heave energy employed as this is the component of energy which displaces waste rock. Strain energy on the other hand causes rock to crush and pulverize particularly adjacent to bore holes. The less energy spent creating new rock surfaces and the more energy providing heave the more successful will be the muck pile movement.

There are a wide variety of explosives on the market today. The optimum explosive for casting in most instances is ANFO. ANFO provides comparatively high heave energy (HE) to strain energy (SE) ratios compared to emulsions, heavy ANFO's and watergel products. Furthermore, ANFO is normally the cheapest explosive on a cost per energy unit basis. In some instances lower density ANFO and decoupled charges have been used to maintain bore hole pressures low enough to minimize crushing and maximize heave. Emulsion and watergel products are relatively expensive and have a lower HE:SE ratio. The use of these explosive types is to be minimized to application in wet holes only.

Figure 8 BLAST CAST VARIABLES

VARIABLES	TYPE	COMMENTS
A. Explosive Type	M	Constrained by Products available and whether the holes are dry or wet.
B. Powder Factor/Energy Factor	M	
C. Rock Properties		
1. Density	F	
2. Young's Modulus	F	
3. Discontinuity Characteristics	F	
D. Blast Geometry		
1. Face Height	F	
2. Length to Width Ratios	F/M	May be constrained by short range planning.
3. Depth to Width Ratios	M	Blast depth is fixed, width is variable
E. Blast Hole Geometry		
1. Hole diameter	F	
2. Inclination	M	Constrained by angle drill specifications and availability.
3. Stand off	M	
4. Stemming Heights	M	
F. Blast Pattern Geometry		
1. S/B Ratios	M	
2. Front Row Burden	F/M	Can be modified by angle drilling.
3. Orientation	M	
4. Pattern Size	M	May be constrained by drill resources.
G. Blast Dynamics		
1. Inline Initiation	M	
2. Delay Intervals and scatter	M	
3. Explosive Initiation	M	

F - Fixed (Normally)
M - Manipulable
F/M - Can be fixed or manipulable

The type of explosive used in many instances is controllable. ANFO was not available for most of the duration of the project due to explosive contract constraints but heavy ANFO products were employed wherever possible. In many of the blast casts where holes were wet, magnafrac, an emulsion based explosive, was the only viable alternative.

Figure 9 provides a listing of the explosives available and their relevant properties.

The partitioning of HE and SE can be modified by initiating the explosive in different ways. Ideally the explosive should be continuously side initiated. In this way all of the explosive is detonated within the run up VOD. The slower the detonation the greater the HE:SE ratio. Continuous side initiation can be achieved by using a down line with sufficiently high core load. This effect can be closely approximated by priming with cast primers spaced at 10 hole diameters apart (229<d<381mm). In this way the proportion of explosive detonating with velocities less than the terminal VOD is maximized.

All blast casts were initiated with down the hole delays (period 10 nonels). In general 2 primers and 2 down lines were used with 3 of each in the very deep holes. The bottom primer was located 1 - 2m from the bottom of the charge and the second located 2 - 3m above the first. This configuration of priming did not provide continuous side initiation in the bore hole. The additional efforts required to develop continuous side initiation were not considered reasonable at this stage of testing.

FIGURE 9 AVAILABLE EXPLOSIVES AND RELEVANT ENGINEERING DATA

Explosive Type	Product Type	Strengthening Agent	Weight* Strength	Density (S.G.)	Bulk* Strength	Water Resistance
Heavy ANFO	SuperAN 2495	Increasing emulsion content	95	0.97	109	NIL
	2505		92	1.09	120	NIL
	2510		91	1.15	125	NIL
	2515		90	1.21	130	Poor
Emulsion	Magnafrac 9000	Increasing Aluminum	81	1.225	118	Excellent
	9013		90	1.225	134	Excellent
	9020		113	1.225	164	Excellent
ANFO**	ANFO	Increasing Aluminum	99	0.75	88	Poor
	5% ALANFO		117	0.78	109	Poor
	10% ALANFO		134	0.80	128	Poor
Power Gels**	PGA	Increasing Aluminum	82	1.25	122	Good
	PGB		93	1.25	138	Good

* Relative to Prill Density of 0.75 gm/cm^3.

** products not used after January 1, 1987.

B. Powder Factor/Energy Factors

Energy factors rather than powder factors provide a more useful indication of energy within the blast. Generally the units of weu/BCM are used. Energy factors for blast casts in general are larger than for standard production blasts. Higher energies can be achieved by reducing hole spacing with a given explosive or by maintaining pattern spacing and using higher bulk strength explosives. Reduced pattern spacings and higher bulk strength explosives are often employed together.

The energy factor is perhaps the most readily manipulable variable. It is determined by hole diameter, spacing, burden, stemming heights, and explosive types employed.

C. Rock Properties

Rock density, elastic modulus and discontinuity characteristics are the most important rock properties for blast casting. These variables are obviously fixed for each blasting situation.

When casting waste material the weight of rock mobilized is of more significance than the volume. Explosive pressure exerted over an area for a given increment of time creates an initial velocity on a mass of rock. A less dense waste rock will require less energy to achieve a specified velocity than a more dense material keeping all other variables constant.

Waste materials with higher elastic moduli and higher breaking strain values will absorb less energy in crushing and elastic deformation than waste materials with a lower elastic modulus and lower breaking strain. Energy absorbed in crushing and elastic deformation is lost from doing useful work. Explosives generating high bore hole pressures can be used if these stiffness and strength parameters are large.

Perhaps the most important rock property is the discontinuity characteristics of the waste rock. The frequency, dilation, persistence and orientation of joints, shears and bedding planes are all important.

Frequent discontinuity occurrences increase the number of channels for premature loss of explosive energy. The amount of dilation across discontinuities affects the rate at which energy is lost particularly for major discontinuities which are persistent to the free face. Joints oriented parallel to the free face enable rapid development of the interhole pressurized plane implying spacings can be increased for a given burden. However if such discontinuities are persistent to the collar of the hole gases could be rapidly vented into the atmosphere reducing the pressure build up and the time over which the pressure is exerted on the plane of waste rock. This in turn would reduce the initial face velocities achieved.

Rock structure is an uncontrollable variable and varies considerably from area to area. Other than stating structure strike and dip, little information on other rock characteristics has been collected.

D. Blast Geometry

1. Face Height

There is an advantage to apply blast casting to the highest practical face height. Higher face heights result in longer flight periods and therefore further travel distances. Furthermore, a larger percentage of the burden is acted upon directly by explosive gases for a given stemming length.

2. Blast Width

The blast dimension perpendicular to the free face is the blast width. The largest proportion of material is cast from the front row of the blast. Subsequent rows in the blast will cast progressively less. As stated previously test blasts were not totally blast casts in most instances. In most cases a reasonable width at the front of the blast was designed for casting and the remainder was designed as a conventional production blast. The optimum depth to width ratios for blast casting is a variable to be determined through testing. Experience at one mine indicated depth to width ratios ranging from 0.7 to 1.0 are reasonable and that the relationship between percentage cast and D/W ratio is linear[1].

3. Blast Length

The length to width ratio of the blast cast should be maximized in order to minimize the negative influences of the fixed ground on either end of the blast. Clearly the stationary ground on either end of the blast tends to restrict the movement of material within the blast cast immediately adjacent to the last row blasted (LRB). The longer the blast the less volume will be subjected to these restrictive forces.

Blast length and blast width are easily manipulable in most cases, but nevertheless subject to production constraints. Blast depth will be a fixed variable in most instances.

E. Blast Hole Geometry

1. Hole diameter

The optimum hole diameter for blast casting is normally smaller than the hole diameter which costs the minimum per BCM of waste drilled. Large hole diameters have longer stemming heights which effectively decrease the percentage of the burden acted on directly by the explosive gases. Test shots employed 270mm and 254mm diameter holes. This diameter range is considered reasonable for blast casting.

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[1] Anthony G. Grippo, "How to Get More Cast per Blast", Coal Age (Dec. 1984), p. 63.

2. Hole Inclination

Angle hole capability assists blast casting in two (2) ways. It has been stated that the deeper the hole for blast casting the better. However even with relatively steep face angles of 70°, heights in excess of 15m result in excessive burdens for vertical holes. Angle holes enable front row burdens to be reasonable for high face heights. When holes can be angled parallel to the face angle the burden is constant along the entire hole length facilitating the development of uniform face velocities. A secondary advantage to angle holes is that they enable explosive forces to act with an upward component which imparts an upward component to initial face velocities facilitating a longer flight time for the waste rock. Considering projectile motion aspects only, the optimal orientation would be 45°. This is impractical from a drilling and loading viewpoint, and would not provide a uniform burden on the front row of holes when face angles are steeper than 45°. In blasting situations where waste is being cast off a flatly dipping seam there would be less coal seam dilution and loss if there was an upward component to the waste movement rather than the waste being pushed directly along the plane of the seam.

Four (4) of the test blasts employed angle hole drilling. The maximum angle used was 20°. No loading problems occurred when loading 20° holes. The likelihood of caving holes increases with angle holes particularly when the collars are in overburden or rehandle material. Drilling angle holes from drill ramps dozed on original ground can be difficult. Ramps have to be wide enough to enable the drill to nose out onto the crest. Wide ramps imply increased thicknesses of overburden material along the outside edge. This can create a caving hole problem as well as a stability concern. The effects of orientation error when drilling angle holes can be severe. Only general guidelines were given to drillers in this regard. Quantitative orientation accuracy is unknown.

When meshing the angle holes of a blast cast to vertical holes of a production blast the collars of the rows of holes come close together. In these situations several holes had to be left unloaded over a period of time until subsequent rows of holes had been drilled off.

3. Standoff Distances

The standoff distance applies to situations where waste is being blasted off a coal seam. The standoff distance is the distance from the bottom of the charge to the top of the coal seam. For standard production blasts the standoff distance is commonly 0.6m. This is increased for blast casts because of the greater amount of shearing motion adjacent to the coal seam.

4. Stemming Heights

The smaller the stemming height the larger the proportion of the hole burden will be subjected to detonating explosive pressures and this implies the development of a larger inter hole pressure plane. As stemming lengths become too short blast hole energy is ejected before the interhole split is formed and fully pressurized. To provide the best confining affects per unit length, stemming material should be granular in nature. Stemming height is easy to change and therefore is considered a controllable variable. Stemming heights for each pattern were decided assuming the collar was in solid rock. In many instances, particularly with outside edges collars were increased if significant thicknesses of rehandle or overburden were present.

F. Blast Pattern Geometry

1. Burden to Spacing Ratios

Hole spacing is defined as the distance between holes which are initiated simultaneously. The burden distance is the effective width of rock in front each hole which the hole must effectively detach. This distance is measured parallel to the blast direction. The optimum burden to spacing ratio will be achieved when the explosive pressures are exerted over the maximum interhole area. For ratios less than the optimum, pressures exit from the face through newly formed cracks before the interhole split is fully formed and pressurized. The optimum S/B ratio is 2.4 for homogeneous rock. Actual S/B ratios must reflect the likelihood of errors in actual collar locations, irregularities in free faces, and discontinuities within the rock mass as explained earlier. In general, S/B equal to 2 is reasonable to account for such imperfections.

2. Pattern Spacing

Determination of actual burden and spacing values given the ratio between the two is selected to give the desired powder factor. Efforts should be made to ensure burdens of the front row of holes are acceptable. If the front row of holes fails to cast effectively because of excessive burden the entire blast cast will be adversely affected.

3. Pattern Orientation

For best results the pattern must be oriented so that the blast direction is perpendicular to the free face.

G. Blast Dynamics

Cast blasts should be initiated line by line. Delay intervals between rows of holes should be the maximum tolerable without causing cutoffs or missholes.

In order to obtain maximum benefits from stress wave superposition between holes, all holes on each line should be initiated simultaneously. This implies that safety lines and firing lines employ the same surface delays in order to minimize the effect of delay period scatter. Ground vibrations increase with the charge weight per delay potentially causing stability problems. This was not a concern for this test work so all holes on each line were initiated simultaneously.

When down the hole delays are used the scatter can be appreciable in real time. Scatter affects can be minimized by using two (2) or more initiators per hole. Furthermore, because delay scatter is a percentage of the delay period the minimum down the hole delay period consistent with blast integrity should be used.

2.2 METHODS OF BLAST CAST ANALYSIS

The previous discussion highlighted the need for presenting fundamental results which are void of site specific factors. When site specific information has been factored in it must be clearly understood. It is equally important to describe in detail the methods used in the analysis.

Each blast cast is analyzed within a separate blast report. All relevant data necessary to analyze the blast is summarized within each report, (2.3.1-2.3.8) as such, each report is an independent entity.

All blast reports are organized in a similar manner so that valid comparisons can be made between different blasts. Each report has an Introduction, a General Information section, and an Analysis section.

The introduction states the location and the objective of the blast as well as a definition of volume cast. The definition of volumes cast are directly linked to the mine plan so that economics can be easily applied. Providing a statement of percentage cast is meaningless without reference to a definition.

The general information section specifies drilling and loading details, accessories employed, initiation details, and basic geologic information. Documentation sources are also listed. Technical or practical difficulties which were noticed are also discussed in this section and a general statement about deviations is also made.

The analysis section develops an incremental cost and benefit of the blast in generalized terms. Costs are generalized so that various values of unit explosive and drilling costs can be easily used to evaluate total costs. Likewise the benefit of casting a unit volume of waste according to the definition made for each blast is formulated in a generalized way.

The general form of the cost/benefit analysis is as follows:

Net Value, NV = Incremental Benefit - Incremental Cost

Incremental Cost, IC = $C_1U_1 + C_2U_2 + \ldots + C_nU_n$

Where: $C_1, C_2, \ldots C_n$ represent incremental cost items (ie. kilograms of various explosive types, drill meters, etc.) at unit costs of U_1, $U_2, \ldots, U_n$. They are determined by subtracting from actual quantities used the qunatitites which would have been used in a standard production shot.

Incremental Benefit, IB = $(Vc_1U_1 + Vc_2U_2 + \ldots + Vc_nU_n) - (Vpc_1U_1 + Vpc_2U_2 + \ldots + Vpc_nU_n) = (Vc_1 - Vpc_1)U_1 + (Vc_2 - Vpc_2)U_2 + Vc_n - Vpc_n)U_n$

Where: $Vc_1, Vc_2, \ldots Vc_n$ represent cast volumes (eliminating equipment handling) at unit savings of $U_1, U_2, \ldots U_n$ and $Vpc_1, Vpc_2, \ldots Vpc_n$ represent estimated production cast volumes at unit savings of $Upc_1, Upc_2, \ldots Upc_n$.

Estimated Production Cast Volume, Vpc = Vt * %Vpc

Where: %Vcp = estimated percent cast in production blast base case/100.

DETERMINATION OF VOLUMES CAST: OPEN AND CLOSED BLAST CASTS

In some instances the volume of material cast could not be deduced directly by surveying the volume of muck considered to be cast. For example when casting material off of a steep hillside it would be impractical to attempt to survey volumes of material landing in the valley bottom and veneering the hillside. A value for volumes cast can be inferred in these instances by comparing the volume of material blasted in BCM's to the volume of LCM's remaining after the shot, the difference when calculated in consistent volumetric units is the amount cast.

This method of calculating volumes cast is sensitive to the swell factor assumed which is necessary to convert a volume of insitu rock to a volume of blasted rock. In order to maintain generality and provide a means for a sensitivity analysis, the swell factor in these instances is left as a variable. Where swell factors are applied to unsucessfully casted waste volumes in order to estimate volumes cast, the type of blast cast is defined as open.

There is minimum inaccuracy of assigning a swell factor of 1.3 to volumes of waste cast. However the waste remaining that has not effectively been cast likely has a swell factor lower than 1.3. The swell factor of waste in place likely varies from approximately 1.3 near the top surface to approximately 1.10 in regions near the bottom and back of the blast cast.

When there is access to survey the casted waste pile, volumes cast in loose cubic meters can be accurately determined. As stated above, a swell factor can be accurately applied to this material to convert to bank cubic meters. These types of blasts are defined as closed blast casts.

Some of the trial blasts, particularly in Lake Pit were shot with significant thickness of soil type overburden against the free face. In these instances overburden volumes were calculated separately. Benefits in casting this material were present so overburden cast volumes were defined. Although overburden could be counted as cast, powder factors and energy factors are derived for bank cubic meters of rock even though a portion of this explosive energy was imparted on overburden material. During the blasting operation the swell of overburden is assumed to be negligible. Dozed overburden and blasted overburden volumes are converted to BCM's by applying a swell factor of 1.25.

Many of the blast casts were shot along with adjacent production blasts. In an ideal situation the blast cast could be shot by itself followed by the adjacent production blast at a later date. Although this would assist in the technical analysis, the constraints to mine operations would be excessive. When combining blast casting and production blasting it is often uncertain how much of the casted waste originated in the production shot. In the extreme case, consider an instance where only a single outside row of a production shot is loaded for a blast cast. A very large percentage of this waste would be cast and a significant amount of waste from second and third was rows could be effectively cast as well. This affect is significant for blast casts with large D/W ratios.

The convention adopted in this study is that percentage cast is the cast volume over the volume of waste drilled and loaded to be cast. With an adjacent production pattern this percentage could in theory exceed 100% as previously stated.

In standard production shots volumes are calculated according to the last row blasted (LRB) limits plus the blast influence limit. Cast blast volumes in this report were calculated to the last row of holes drilled and loaded for casting. The blast influence was not added in these situations.

D/W RATIO

The literature indicates that the D/W ratio is a useful parameter to relate to percentage cast. For this study the depth dimension is defined as the depth of the front row hole. Where blasting to bench level it is the depth less subgrade; where blasting to top of coal it is the depth to top of coal. The width dimension is the distance from the back of the blast cast to the rock/rehandle contact on the front face at the vertical location of D/2.

Where significant quantities of overburden are known to exist in front of the free face D/W ratios can also be devised for widths extending to the outside surface of the overburden. D/W ratios calculated in this way are clearly indicated. The depth of the last row of holes in the blast cast is not a factor in this ratio as defined.

FRONT ROW TOE BURDEN

Front row toe burden is the horizontal distance from the bottom of the hole to the rock/air or rock/rehandle contact on the free face. Averaging can be done by planimetering the area between the collar of the front row holes and the toe line and dividing by the blast length. Averaging can also be done by the section method. The section method is used exclusively when angle holes have been drilled. Where significant thicknesses of overburden exist against the free face the effective burden is further increased. The additional burden that is effectively mobilized in these situations in terms of equivalent intact rock burden is unknown.

ROCK/REHANDLE INTERSECTION

If the thickness of rehandle or OB on a face is unknown the entire cross section out to the face is assumed to be rock.

POWDER FACTORS (PF)/ENERGY FACTORS (EF)

Powder factors are determined by adding up the total mass of each type of explosive within the holes in the blast cast and dividing by the volume of waste rock devised for the corresponding zone. Energy factors are determined by suming the products of mass of each explosive type and the RWS in Figure 9 and dividing by the determined volume. These factors were calculated on the basis of pump sheet records rather than loading instruction records.

CROSS SECTIONS

Pre-blast cross sections were drawn by compositing the best available information from pit mylar pick ups, topographic maps, and clinometer readings from crests. Post blast section information was taken from survey pick ups.

2.3 BLAST CAST REPORTS

2.3.1 BRY #1 Blast Report

Introduction

The objective of this shot is to explore the feasibility of casting waste off of a steeply inclined coal seam. Although all the waste cast would still have to be hauled, there is potential to reduce the cost of effectively cleaning the seam by reducing the dozer intensive notch cut. For the purposes of this analysis the volume cast is the volume removed from the dozer notch area.

Secondary benefits include the utilization of a local free face to assist in fragmentation. Furthermore, the potential of reducing coal loss and dilution by leaving the footwall wedge unblasted may also lead to significant benefits.

General Design Information

Figure 10 shows a plan and sectional view illustrating pertinent details. The waste on top of 13u seam was loaded for blast casting. The waste between 13u and 13 seam was loaded conventionally.

The deviation between design and actual collar locations was not significant.

This blast cast was not filmed with a high speed camera or a video because there was no vantage point available.

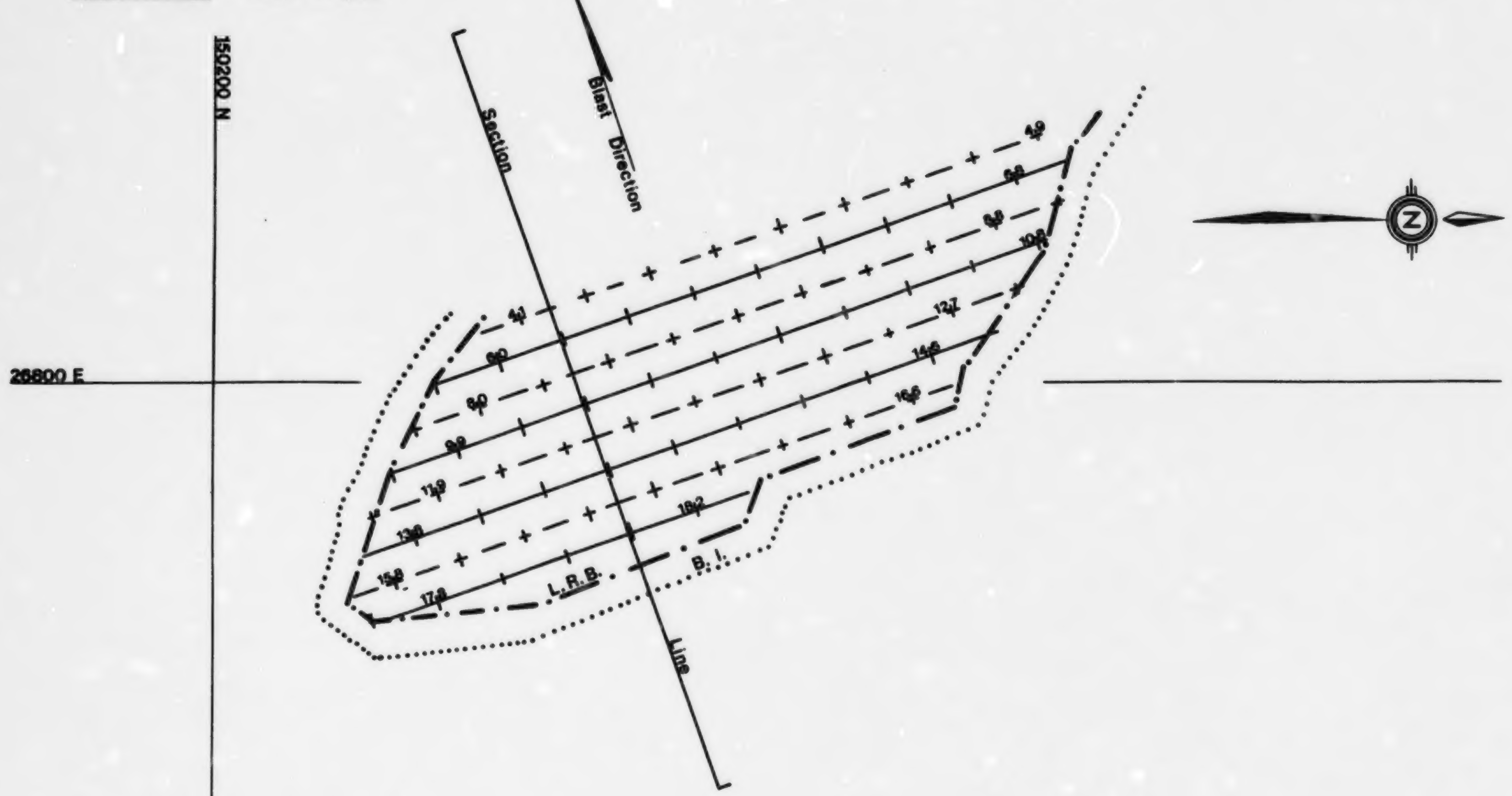
Pattern Design Information

Drilling

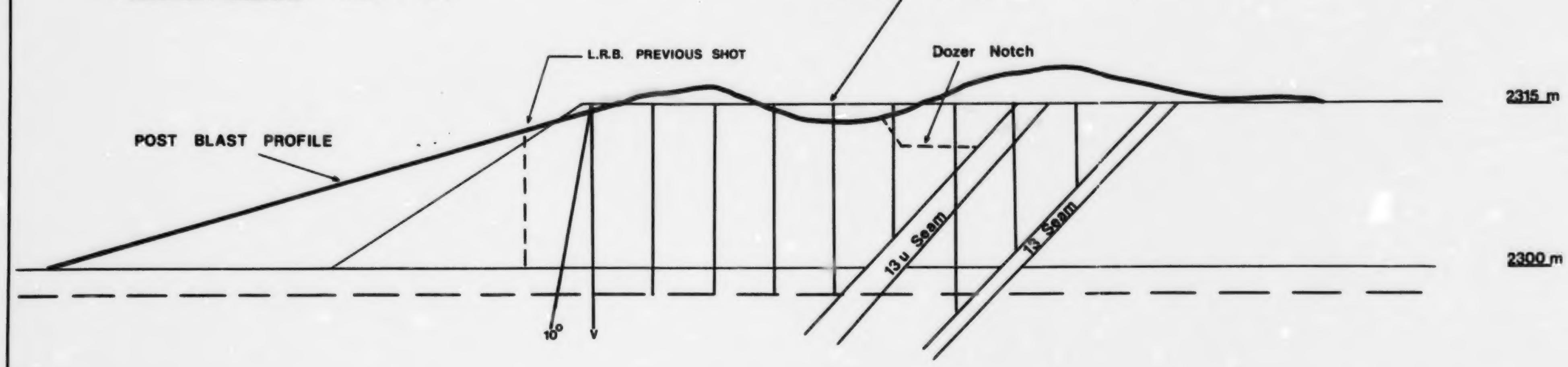
- Hole Diameter - 254 mm
- Pattern Dimensions - 5.5m x 11m staggered; B/S = 0.5
- Angle Drilling Details - every other hole along crest row inclined

Figure 10 BRY Blast Cast

PLAN VIEW Scale 1 : 1,000


150200 N
26800 E
Section
Line
Blast Direction
4.9
6.8
8.8
10.8
12.7
14.6
16.6
4.1
6.0
8.0
9.9
11.9
13.8
15.8
17.8
18.2
L.R.B.
B. I.
Z

CROSS SECTION Scale 1 : 500


PRE BLAST PROFILE
L.R.B. PREVIOUS SHOT
Dozer Notch
POST BLAST PROFILE
2315 m
2300 m
13 u Seam
13 Seam
10°
V

at 10°.

- Average D/W =0.3 - 0.43;   Average L/W = 3.4
- Average Front Row Toe Burden = 3m unbroken rock + 13m broken rock
- Adverse LRB orientation - none
- General - crest holes half spaced. Alternate holes only are angled at 10°. All crest holes drilled and loaded 3m extra.

Loading Details

- Design Loads and Collars
  1. crest holes - 10° Angle - 3m toe load 10% ALANFO
     - collar 6.7m with 5% ALANFO
     - Vertical - 6.1m toe load 10% ALANFO only.
  2. remaining holes - Fill to bench plus subgrade or first coal and load to 6.7m collar with 5% ALANFO.
  3. parting between 13 and 13u loaded conventionally.
- Actual PF/EF - 0.46 kg/BCM; 58.5 weu/BCM
- Percentage ANFO Type Explosive - 98%.
- Location of Wet Hole Explosive - back rows on top of 13u.

Initiation

- DTH delays - period 10 Nonels (2 per hole)
- surface delays - 100ms (blast cast portion only)
- blast direction - perpendicular to face, down dip.

Geology

- strata dipping at approximately 45° to the west. Strata consists of interbedded sandstones, siltstones, mudstones, and coal stringers.

General Comments

- A complex sequencing was necessary in order to shoot the 13u hangingwall waste down dip and the parting between 13u and 13 seam up dip. Holes drilled through 13u had top charges primed with period 14 Nonels (650ms) and bottom charges primed with period 10 Nonels. In this way the parting between 13u and 13 seam would be shot before the hangingwall waste.

Analysis

The cross section shown in Figure 10 illustrates pre blast and post blast profiles. Figure 11 shows photos taken of the front face after the shot. Figure 12 represents a summary of incremental costs and the benefits associated with the blast. The incremental costs are attributable to the extra explosives, accessories, and drilling. In terms of reducing dozer requirements for slot construction there were no significant benefits over a conventionally blasted situation. It is likely that benefits would improve if d/w ratios were increased or powder factors increased.

Figure 12 BRY #1 Incremental Costs and Benefits

Incremental Costs	
$19{,}936\ U_{5\%} + 4{,}016\ U_{10\%} + 345\ U_{PGB} - 345\ U_{PGA} - 20{,}232\ U_{ANFO}$	} explosives
$+\ 20(U_N + U_P)$	} accessories
$+\ 230\ U_D$	} drilling
Incremental Benefits Nil - no significant reduction in dozer notch construction requirements.	

More test blasts are required in order to assess the benefits of blast casting off of steeply inclined coal seams. This technique may be more practical when conditions are such that only one seam is influenced by the blast. Blasting the parting between 13u and 13 up dip and the hangingwall waste on 13u down dip resulted in some excessive 13u seam damage probably due to the choking of charges loaded within the parting. With conventional blasting the 13 seam footwall would have been blasted providing some lateral relief for the parting waste between 13 and 13u seam.

Photo of post blast face as the south end of the pattern (looking north)

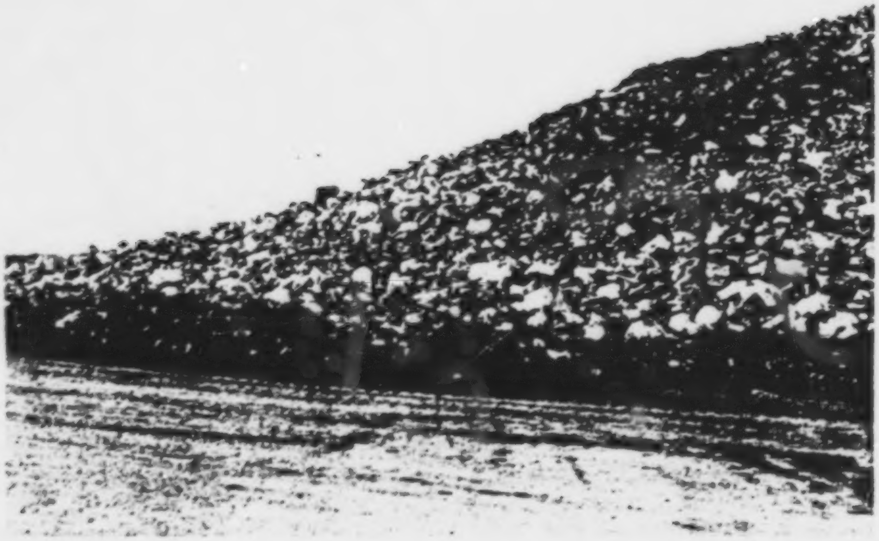


Photo of post blast face in background at north end of pattern. Note the contrast to the unblasted face in the foreground.

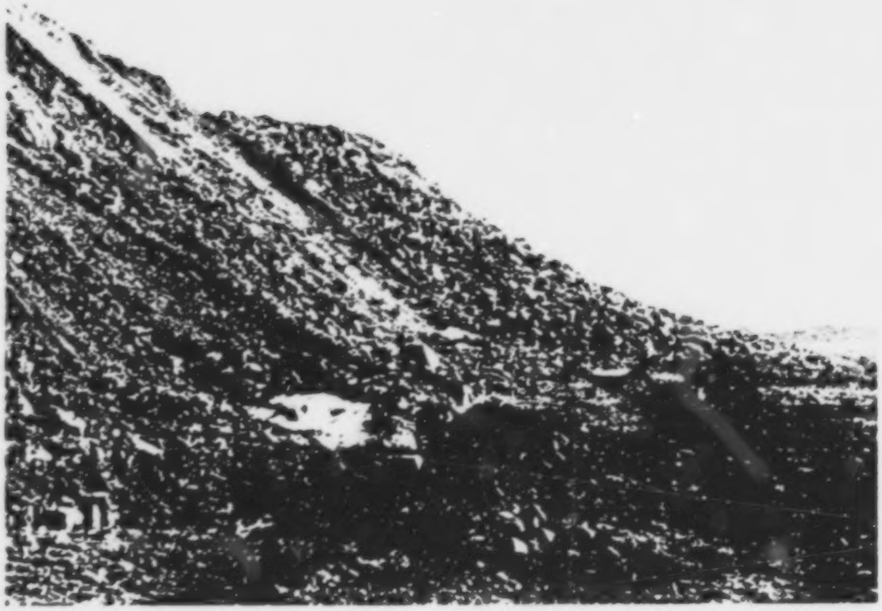


Figure 11 BRY #1 Post Blast Photographs

2.3.2 BRA #2 Blast Report

Introduction

The objective of this blast cast was to cast waste off the ridge into Brownie Valley. Waste cast lower than the 2,060m elevation would not have to be subsequently rehandled. Waste Cast below the 2195m Bench but landing above the design pit limit elevation would not have to be moved for a long period of time (ie mining deferred).

General Information

Figure 13 shows a plan and 2 cross sections through BRA #2.

An attempt was made to video tape the shot but the video camera failed probably as a result of the cold weather. A collection of before and after still photographs were taken and are illustrated in Figure 14.

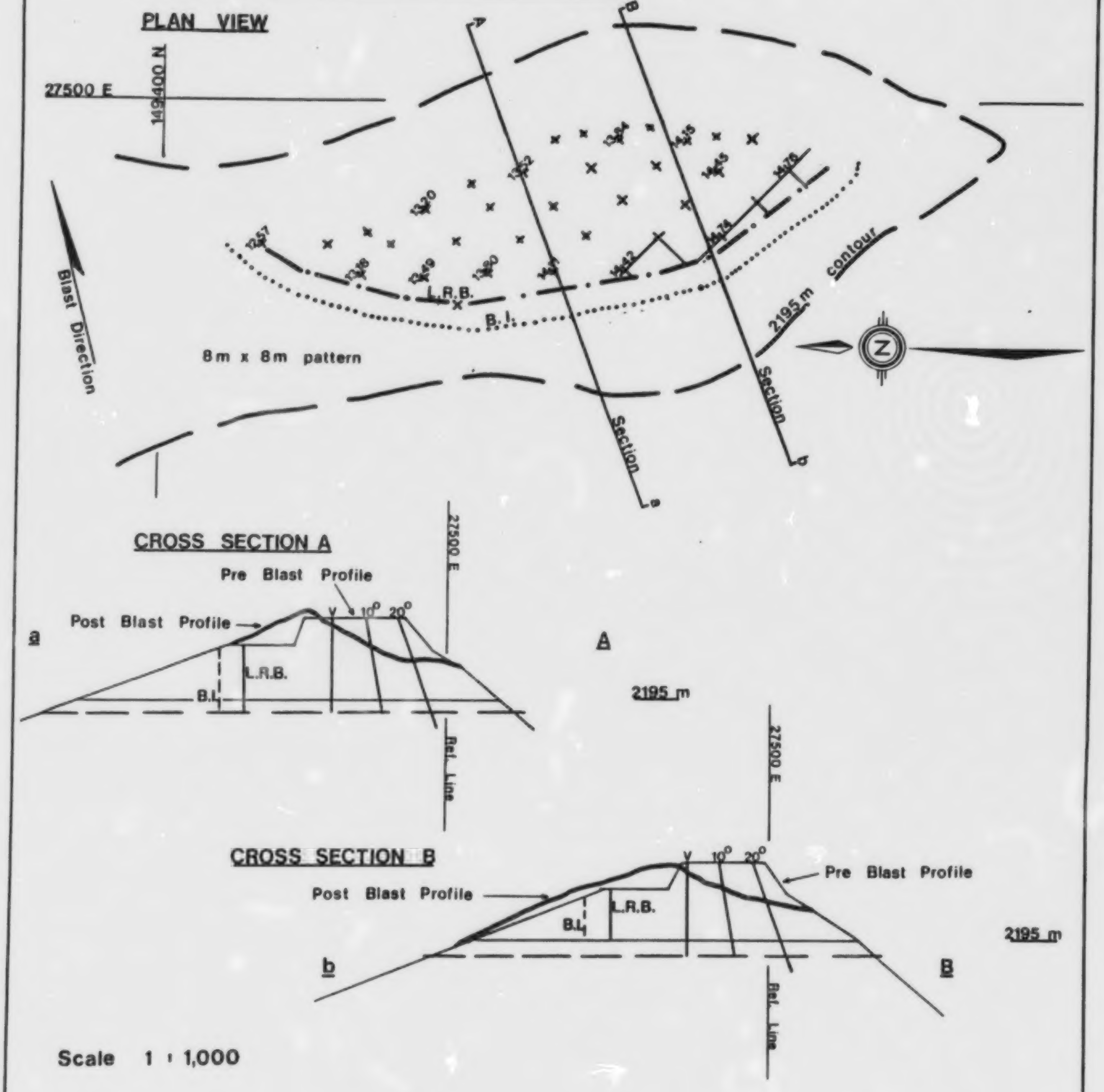
Drill ramps had to be constructed on the rock knob in order to provide a drill platform. As a result of the platform shape and size, the variance between design and actual hole locations was severe. It is estimated that variance ranged from 0 - 3m with a mean of 2m. The error in orientation of the drill when drilling angle holes is estimated at $\pm 15^{\circ}$ in azimuth.

Pattern Design Information

Drilling

- Hole Diameter - 254mm
- Pattern Dimensions - 8m x 8m square (5.6m x 11.3m staggered); B/S = 0.5
- Angle Drilling Details - crest holes at 20°, 2nd row at 10°.
- Average D/W = .38; Average L/W = 2.75
- Average Front Row Toe Burden = 11.2m
- Adverse LRB orientation - none
- General - east side crest holes were drilled on half spacing and

Figure 13 BRA 2 Blast Cast


PLAN VIEW
27500 E
149400 N
A
B
1257
1318
1320
1349
1352
1380
1384
1411
1415
1442
1445
1474
1476
L.R.B.
B.I.
8m x 8m pattern
2195 m contour
Blast Direction
Section
Section
a
b
Z
CROSS SECTION A
Pre Blast Profile
Post Blast Profile
a
V
10°
20°
L.R.B.
B.I.
27500 E
Ref. Line
A
2195 m
CROSS SECTION B
Post Blast Profile
Pre Blast Profile
V
10°
20°
B.I.
L.R.B.
27500 E
2195 m
b
B
Ref. Line
Scale 1 : 1,000

loaded 3m extra. The drilling was done from 2 narrow ramps. Collar locations were moderately off pattern. The orientation error of the angle holes is estimated at ±15° in azimuth.

Loading Details

- Design Loads and Collars - all holes 4.6m toe load - 10% ALANFO, loaded to 5.5m collar with 5% ALANFO.
- Actual PF/EF - 0.74 kg/BCM; 91 weu/BCM
- Percentage ANFO type explosive - 100%
- Location of Wet Hole Explosive - N/A

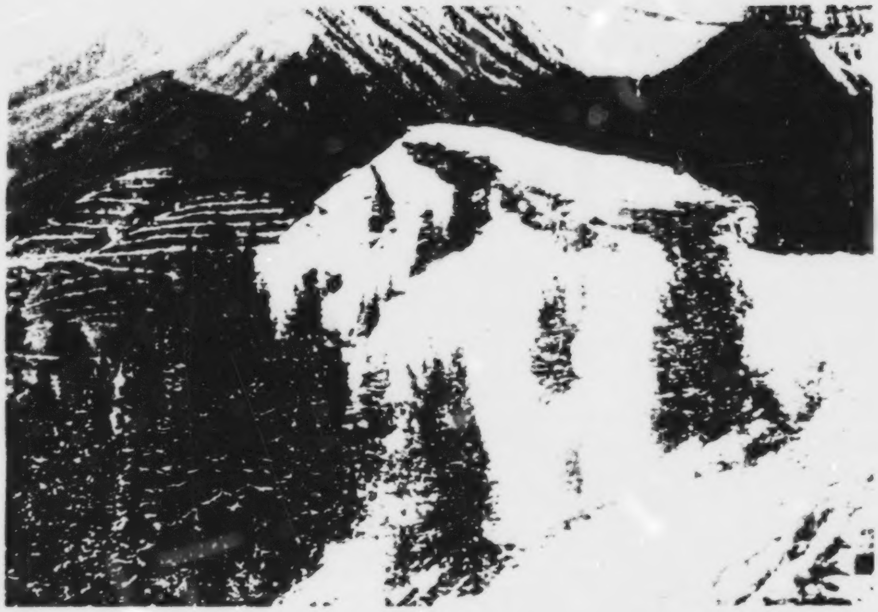
Initiation

- DTH delays - period 10 Nonels 2 per hole
- surface delays - 100 ms
- blast direction - approximately perpendicular to free face - straight up dip.

Geology

- massive sandstone above 9sm, dipping at 30 - 45° at a direction 180° from the blast direction.

Pre Blast photo of BRA #2. Drilling done on 2 narrow ramps. Attention was given to angle hole orientation in view of the narrow ramp. Also note the steep topographic slopes. Except for the near vertical segment near the crest topographic slope angles approach 40° on the front face.



Post Blast Photo. For scale note the 2 people walking on the highest point of the profile.

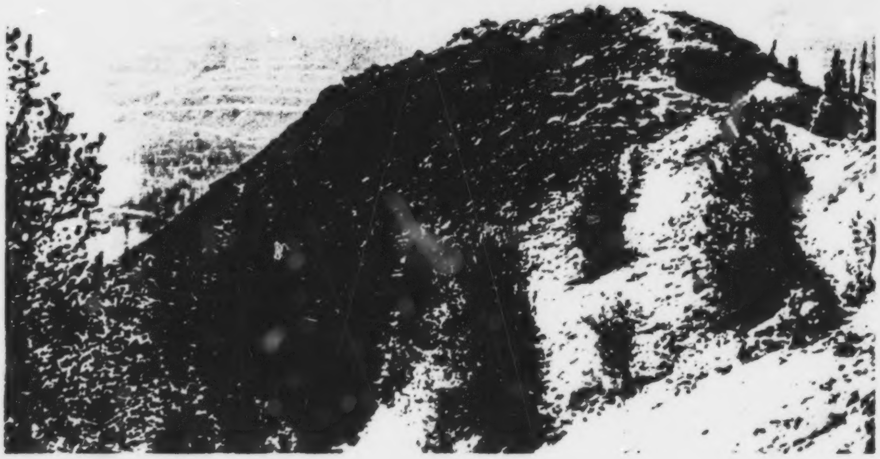


Figure 14 BRA #2 Pre and Post Blast Photographs

Analysis

BRA #2 was an open type of blast cast as access to pickup all the blasted waste was not available. As a result cast volumes must necessarily be stated according to an assumed swell factor.

Cast volumes were determined from four sections, two of which are illustrated in Figure 13. Figure 15 shows a summary of the data collected. Assuming a swell factor of 1.3, 8,050 BCM's were cast below the 2195m elevation. This volume represents a 25.3% cast.

Figure 15 BRA #2 Blast Results

Section	D/W	Pre Blast Area BSM	Post Blast Area LSM	Area Cast BSM	% Cast	
					S=1.2	S=1.3
1	0.40	353	349	353-349/S	17.6	24
2	0.38	490	435	490-435/S	25.9	32
3	0.33	511	532	511-532/S	13.2	20
4	0.41	440	419	440-419/S	20.6	27

VOLUME CAST, Vc = 31,815 - 30,888/S FRACTION CAST, % Vc = 1 - 0.97/S

For S = 1.2 Vc = 6,075 BCM's Percent Cast = 100 * %Vc = 19.1

For S = 1.3 Vc = 8,050 BCM's Percent Cast = 100 * %Vc = 25.3

Additional drilling costs were incurred as a result of half spacing the crest holes. A standard production size grid was used so that no additional costs resulted from grid spacing changes. The additional cost of using a contract drill was not included in the analysis.

Incremental explosives costs resulted from additional holes loaded, reduced collars, and the higher strength explosive used in the pattern. The additional number of holes along the crest resulted in a minor incremental cost of accessories.

Figure 16 shows a summary of the incremental costs and benefits for the blast cast in generalized terms. The volume cast as a function of swell factor and the estimated unit cost of Truck/Shovel mining (side casting in this case) applied to this volume derives the total benefit.

Figure 16 BRA #2 Incremental Costs and Benefits

Incremental Cost

$1{,}561\ U_{10\%} + 6{,}487\ U_{5\%} - 5{,}088\ U_{ANFO}$ } explosives

$+ 14\ (U_P + U_N)$ } accessories

$+ 149\ U_D$ } drilling

Incremental Benefits

$= (Vc - Vpc)\ U_{TS}$

$= (Vc - Vt * \%\ Vpc)\ U_{TS}$

$= [(31{,}815 - 30{,}888/S) - (31{,}815 * \%\ Vpc)]\ U_{TS}$

$31{,}815\ (1 - 0.97/S - \%\ Vpc)\ U_{TS}$

For $S = 1.2$ and $\%Vpc = 0.05$ estimated

$= 4{,}507\ U_{TS}$

2.3.3 BRA #3 Blast Report

Introduction

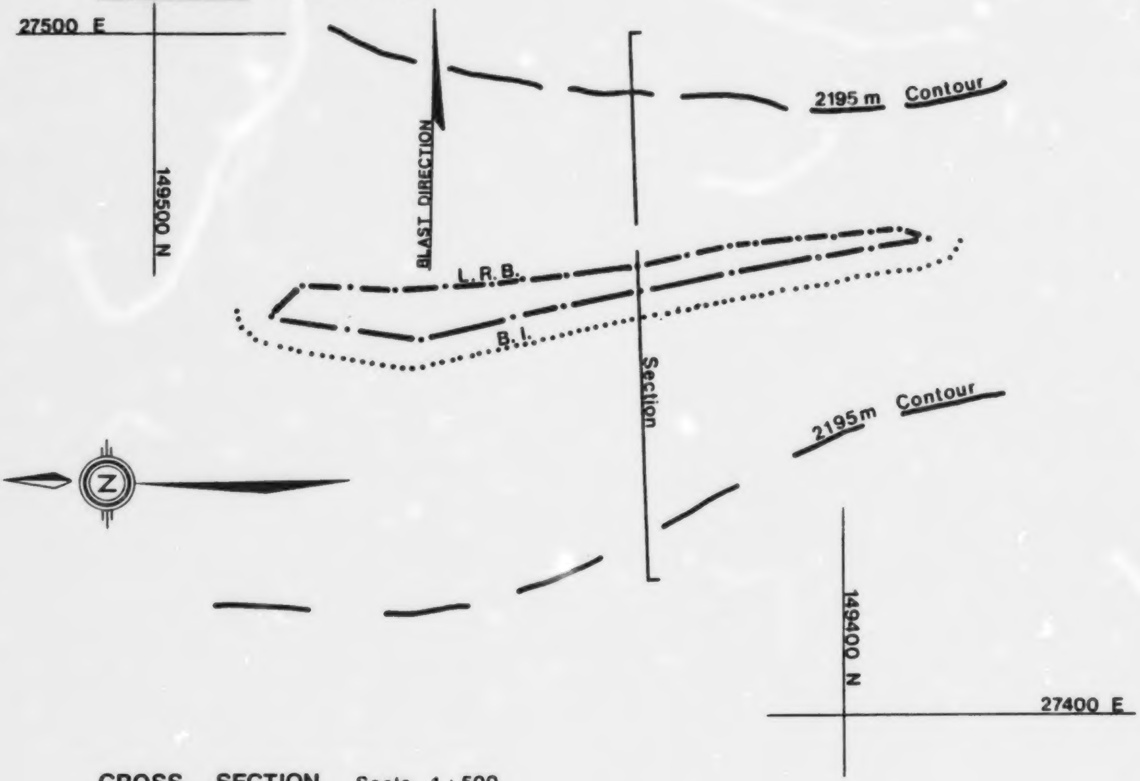
The objective of this blast was identical to BRA #2, to investigate the casting of waste material off topography. As with BRA #2, waste material moved off the 2195 bench is considered cast.

General Information

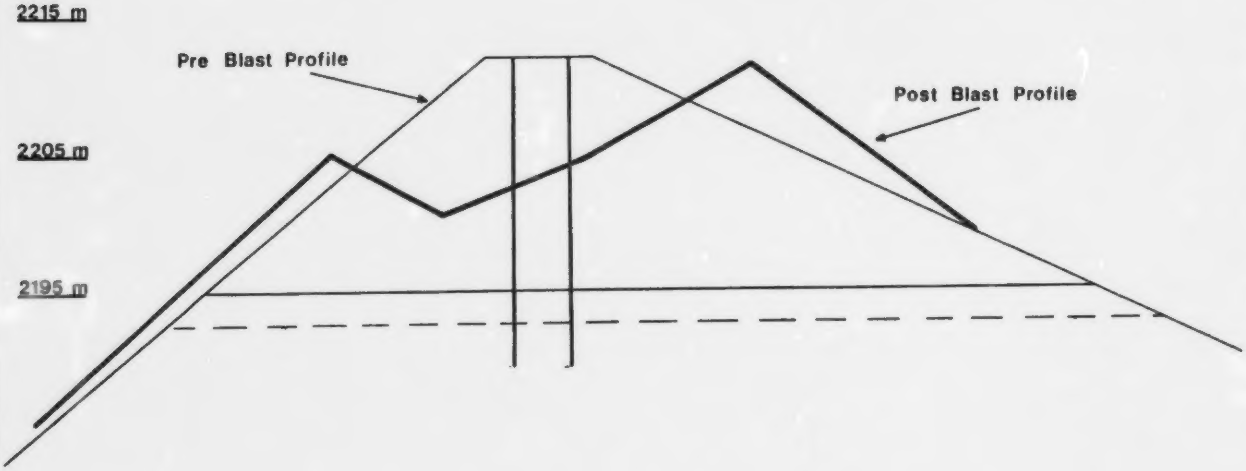
BRA #3 pattern consisted of 2 rows of blast holes aligned along a ridge line with steep topographic slopes on each side. Figure 17 shows a plan view and cross section through the pattern. Although angle drill capabilities were available angle holes could not be drilled because the

Figure 17 BRA 3 Blast Cast

PLAN VIEW Scale 1 : 1,000


27500 E
149500 N
BLAST DIRECTION
2195 m Contour
L.R.B.
B.I.
Section
Z
2195m Contour
149400 N
27400 E

CROSS SECTION Scale 1 : 500


2215 m
2205 m
2195 m
Pre Blast Profile
Post Blast Profile

ridge line was too narrow to nose the drill out to the crest. Although holes were spaced according to a staggered 5.6 x 11.3m staggered pattern configuration, the nature of the drilling platform was such that holes were field fit. A video or high speed film was not taken of the blast however photographs were taken before and after the shot.

Pattern Design Information

Drilling

- Hole Diameter - 254mm
- Pattern Dimensions - 5.6 x 11.3 staggered; B/S = 0.5
- Angle Drilling Details - none
- Average D/W =1.1; Average L/W = 17
- Average Front Row Toe Burden = 17m (an unknown percentage of this would be OB.
- Adverse LRB orientation - none
- General - crest holes drilled 3m extra and were half spaced.

Loading Details

- Design Loads and Collars - 4.6m toe load 10% ALANFO, loaded 5.5m collar with 5% ALANFO.
- Actual PF/EF - 0.73 kg/BCM; 89 Weu/BCM
- Percentage ANFO Type Explosive - 100%
- Location of Wet Hole Explosive - N/A

Initiation

- DTH delays - period 10 Nonels (2 per hole)
- surface delays - 100ms
- blast direction - perpendicular to free face - up dip

Geology

- Massive sandstone dipping at 45°, 180° from blast direction.

General Comments

- The pattern consisted of only 2 rows. After the front row initiated there was likely considerable backward movement because of the close proximity of a free face to the back row.

Analysis

Figure 17 shows a typical pre blast and post blast cross section. Similar to BRA #2 this blast cast is defined as open and therefore volumes cast are functions of the assumed swell factor. The photographs in Figure 18 show pre and post blast surface configurations. Figure 19 shows the relevant analytical information compiled from 3 cross sections. Assuming S=1.3, 39% of the material shot was cast beyond the 2195m elevation.

The incremental costs of BRA #3 include the extra cost of drilling and loading the half spaced holes along the east crest, the cost of high bulk strength explosives, and the cost of raising the collars. A summary of the incremental costs and benefits is shown in Figure 20 along with a summary of the estimated benefits resulting from the blast cast.

2.3.4 CC #2 Blast Report

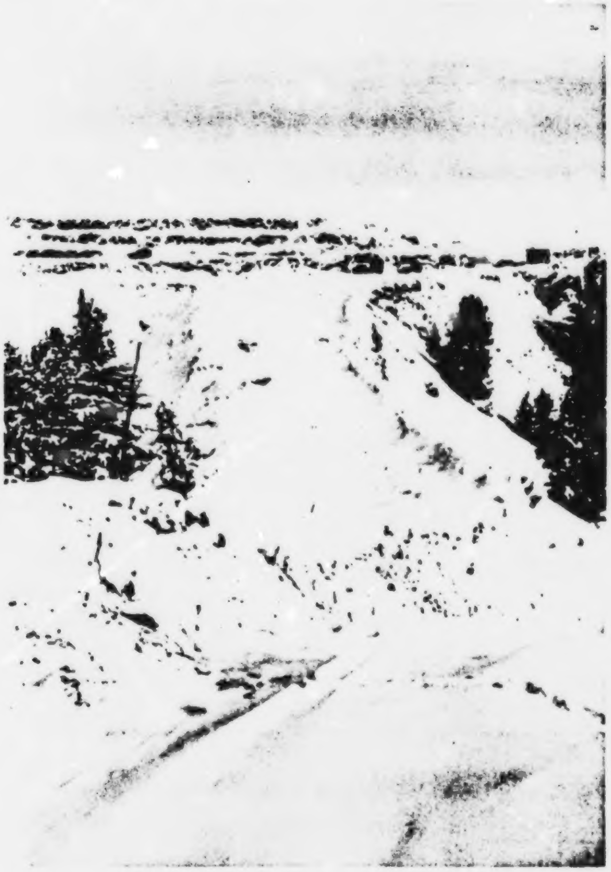
Introduction

The objective of the blast cast was to move waste off the 1805 bench into the adjacent mined out cut. Material cast beyond the 9 seam footwall would not have to be subsequently rehandled. For the purposes of this analysis, material moved off of the 1805m Bench is considered cast.

General Information

Figure 21 shows a plan and two typical cross sections through the blast cast portion of CC #2. The free face represents an old mined cut. Most of the face except in the area of section Aa was veneered with a layer of soil material. The original rock contact profile is unknown.

Pre Blast Photo
looking north
along the loaded
BRA #3 pattern.
Blast direction
is to the right.



Post Blast photo
looking south.
Blast direction
to left. The
notch seen here is
clearly evident
on cross sections.

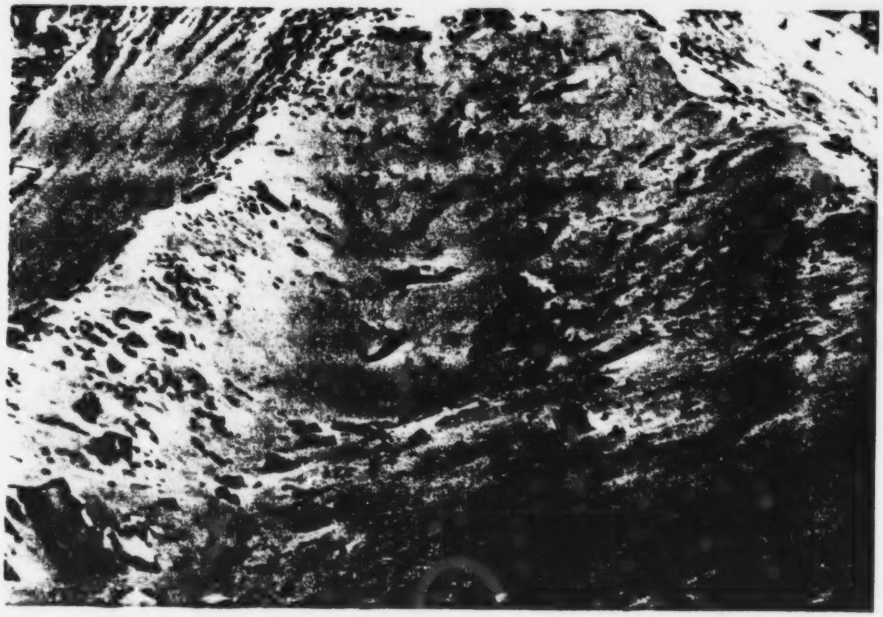


Figure 18 BRA #3 Pre and Post Blast Photographs.

Figure 19 BRA #3 Blast Results

Section	D/W	Pre Blast Area BSM	Post Blast Area LSM	Area Cast BSM	% Cast	
					S=1.2	S=1.3
1	1.1	436	315	436-315/S	40	44
2	1.0	612	538	612-538/S	27	32
3	1.1	511	384	511-384/S	37	42

VOLUME CAST, Vc = 25,598 - 20,264/S FRACTION CAST, % Vc = 1 - 0.79/S

For S = 1.2 Vc = 8,711 BCM's Percent Cast = 100 * % Vc = 34.0

For S = 1.3 Vc = 10,010 BCM's Percent Cast = 100 * % Vc = 39.1

Figure 20 BRA #3 Incremental Costs and Benefits

Incremental Costs

$= 1,955\ U_{10\%} + 8,582\ U_{5\%} - 3,834\ U_{ANFO}$ } explosives

$+ 14\ (U_P + U_N)$ } accessories

$+ 166\ U_D$ } drilling

Incremental Benefits

$= (Vc - Vpc)\ U_{TS}$

$= (Vc - Vt * \%\ Vpc)\ U_{TS}$

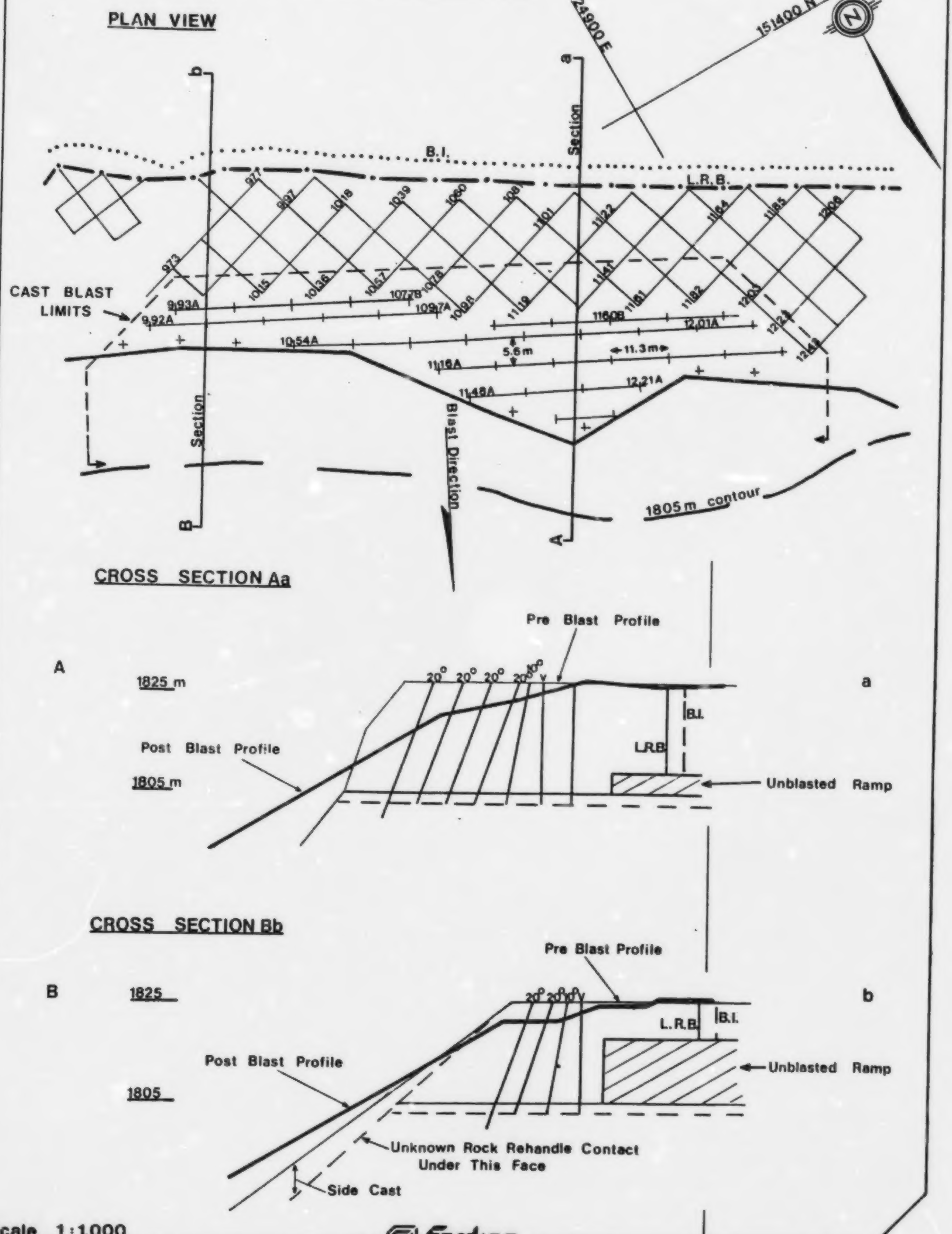
$= [(25,598 - 20,264/S) - (25,598 * \%Vpc)]U_{TS}$

$= 25,598\ (1 - 0.79/S - \%Vpc)U_{TS}$

For S = 1.2 and %Vpc = 0.07 estimated

$= 6,919\ U_{TS}$

Figure 21 CC 2 Blast Cast


PLAN VIEW
24900 E
151400 N
N
Section a
b
B.I.
L.R.B.
971
997
1018
1039
1060
1081
1101
1122
1164
1185
1206
973
1015
1036
1057
1077B
1078
1098
1119
1141
1161
1182
1203
1224
1243
CAST BLAST LIMITS
9.93A
9.92A
1097A
1160B
12.01A
10.54A
5.6m
11.3m
11.16A
11.46A
12.21A
Section
Blast Direction
1805 m contour
B
A
CROSS SECTION Aa
A
a
1825 m
1805 m
20°
Pre Blast Profile
Post Blast Profile
B.I.
L.R.B.
Unblasted Ramp
CROSS SECTION Bb
B
b
1825
1805
20°
Pre Blast Profile
Post Blast Profile
L.R.B.
B.I.
Unblasted Ramp
Unknown Rock Rehandle Contact Under This Face
Side Cast

Scale 1:1,000

fording
COAL LIMITED

The free face extends from the 1825m elevation down to the footwall of 9 seam at approximately 1765m elevation. The bottom of the cut could not be accessed for survey work. CC #2 is therefore considered an open blast cast with results dependent on an assumed swell factor. Figure 22 is a photograph showing the entire free face of the blast cast portion of CC #2.

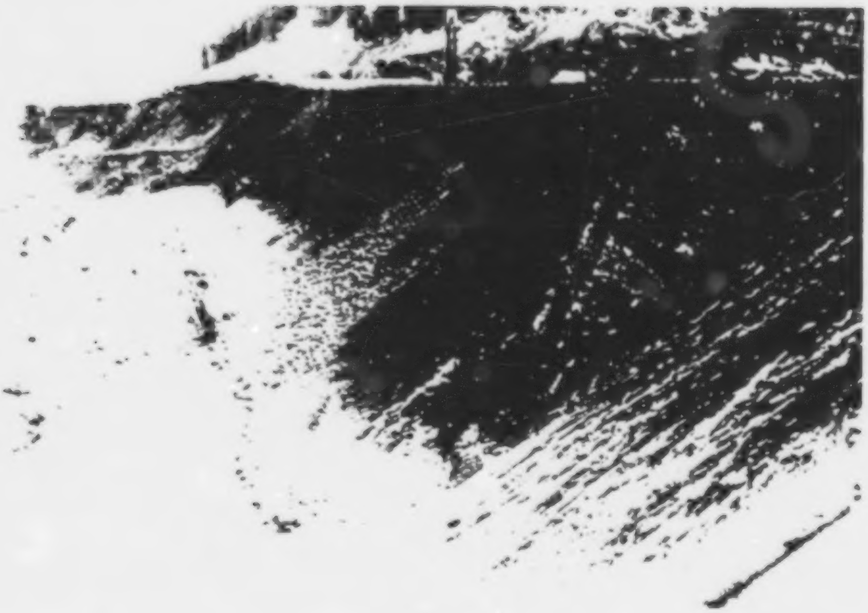
CC #2 was photographed successfully with a high speed camera and with video.

Hole locations were severely off pattern particularly in the rock knob area (Section Aa). Additional holes should have been drilled around some crest portions of the rock knob to overcome excessive toe burden.

Pattern Design Information

Drilling

- Hole Diameter - 270 mm
- Pattern Dimensions - 5.6 x 11.3m; B/S = 0.5
- Angle Drilling Details - 1-4 rows at 20°, 1 10° row to mesh to vertical pattern
- Average D/W = 0.6;   Average L/W = N/A, Production blast on east end.
- Average Front Row Toe Burden - 17m. Burden extremely variable ranging from 10m at section Aa to 20m on either side of this section. In vicinity of section Aa the free face was solid rock. On either side the free face was veneered with an unknown thickness of OB so the effective burden was less than 20m.
- Adverse LRB orientation - none
- General - crest holes drilled and loaded 3m extra. Regular spacing along crest. Remaining holes drilled to 1803m (2m subgrade); approximately 22m deep.



This photo shows the nature of the free face. The blast cast portion of CC #2 ranges from the target in the foreground, past the rock knob to the approximate location of the CIL explosive truck. The nature of the free face is variable ranging from a steeply plunging, exposed rock face in the vicinity of the rock knob to an overburden covered face inclined at 41°. The nature of the rock face behind the overburden material is unknown. The targets suspended on the free face were used for face velocity assessments. Face velocities of 17.8 m/s and 14.5 m/s were attained at the midpoint and toe of the rock face. A face velocity of 10.6 m/sec was attained 15m below the crest on an OB face.

Figure 22 CC #2 Pre Blast Photograph

Loading Details

- Design Loads and Collars - 3m toe load Mag 9020, load to 6.4m collar with Mag 9013. Dry holes - loaded to 6.4m collar with SuperAN 2510.
- Actual PF/EF - 0.92 kg/BCM; 86 weu/BCM
- Percentage Heavy ANFO Explosive - 10.7%.
- Location of Wet Hole Explosive - everywhere except holes at back of blast.

Initiation

- DTH delays - period 10 Nonels (2 per hole)
- surface delays - 100ms. High speed film analysis indicated that the delay between rows was 99.5ms which is in excellent agreement with nominal firing times.
- blast direction - perpendicular to free face. Slightly down dip.

Geology

- Competent moderately massive sandstones and siltstones between 9 and 11 seam.

General Comments - N/A

Analysis

The photographs in Figure 23 provide a good comparison for a visual appraisal of the blast cast. The rock knob has been almost completely cast into the mined out cut.

Figure 21 shows typical pre blast and post blast cross sections. Data derived from these and other sections is shown in Figure 24. As previously discussed this blast cast is an open type therefore results are a function of an assumed swell factor. For a swell factor of 1.2 the volume cast is 26,150 BCM where as for a swell factor of 1.3 the volume cast increases to 30,700 BCM.

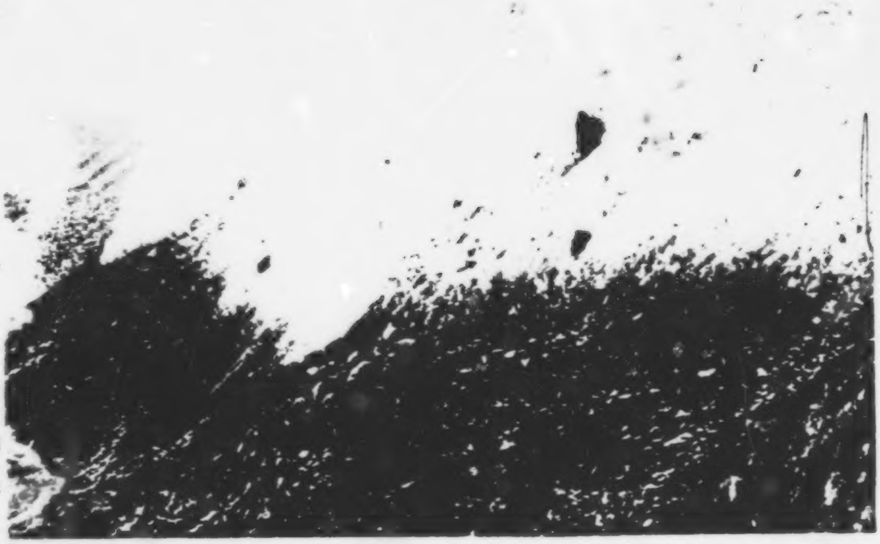
On a section by section comparison, Section 4 had the steepest free face and therefore the smallest toe burden and had the largest percentage cast even though the D/W ratio was relatively small. Considering all sections there does not appear to be a clear relationship between D/W and the percentage cast. Unfortunately various sections have different toe burden and free face conditions which would tend to cloud any such trend.

The incremental costs associated with the CC #2 blast cast arise from reduced collars, reduced pattern spacings and higher bulk strength explosives over a production pattern. The blast cast pattern had an EF = 86 weu/BCM compared with a property average of 50 weu/BCM.

The general expression for incremental costs and benefits based on various component unit costs is shown in Figure 25.

High speed film analysis indicated that face velocities of 17.6 m/s and 14.5 m/s were attained at the midpoint and the toe respectively of the rock face. A face velocity of 10.6 m/s was attained 15m below the crest of a face veneered with an unknown thickness of OB.

Pre Blast Photo of CC #2.



Post Blast Photo of CC #2 from approximately the same location as the photo above. Note that the rock knob is essentially gone and that the overall ground elevation has dropped 2 meters in the blast cast area

Figure 23 Pre and Post Blast Photos of CC #2.

Figure 24 CC #2 Blast Results

Section	D/W	Pre Blast Area bm^2	Post Blast Area Lm^2	Area Cast bm^2	% Cast	
					S=1.2	S=1.3
1	0.67	616	527	616-527/S	29	34
2	0.75	573	521	573-521/S	24	30
3	0.67	653	542	653-542/S	31	36
4	0.53	947	734	947-734/S	35	40
5	0.43	776	676	776-676/S	27	33

VOLUME CAST, Vc = 85,000 - 70,616/S FRACTION CAST, % Vc = 1 - 0.83/S

For S = 1.2 Vc = 26,150 BCM's Percent Cast = 100 * %Vc = 30.8

For S = 1.3 Vc = 30,700 BCM's Percent Cast = 100 * %Vc = 36.1

Figure 25 CC #2 Incremental Costs and Benefits

Incremental Costs

$= 7{,}897\ U_{9020} + 8{,}314\ U_{2510} - 55{,}308\ U_{9013} - 54{,}306\ U_{9000}$ }
} explosives
$- 6{,}106\ U_{2495}$ }

$+ 14\ (U_P + U_N)$ } accessories

$+ 168\ U_D$ } drilling

Incremental Benefits

= (Vc - Vpc) U

= (Vc - Vt * % Vpc) U_{DL}

= [(85,000 - 70,616/S) - (85,000 * %Vpc)]U_{DL}

= 85,000 (1 - 0.83/S - %Vpc)U_{DL}

For S - 1.2 and %Vpc = 0.07 estimated

= 20,203 U_{DL}

2.3.5 - 2.3.8 LAKE PIT CAST BLASTS

There were four (4) blast casts in the Lake Pit area. Three of the L-G pattern blasts were located in Cut 2 with the LJ pattern blast located in Cut 3. The objective of all these blast casts was to cast waste off of the hangingwall of M seam into the mined out cut. For purposes of analysis, volumes cast beyond the L seam mined limit do not have to be rehandled. Waste volumes cast beyond the M seam mined limit have to be rehandled for L seam mining but not for M seam mining. This implies that waste cast beyond these boundaries will have different economic benefits.

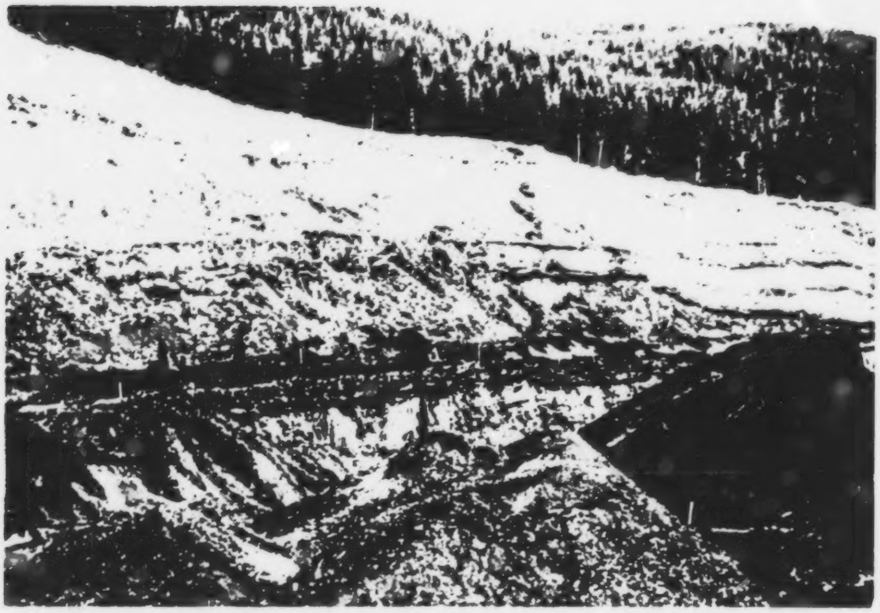
All blast casts in Lake Pit are considered closed blasts. Access to the post blast profile was sufficient to deduce a cast volume that was independent of the swell factor of the blasted but unsucessfully cast waste rock.

All L-G blast casts necessitated the construction of drill ramps in OB material in order to provide a level drilling platform. Large quantities of OB had to be dozed over the free face before ramps could be cut. This was necessary to reduce the redrill problem which would likely become severe if holes were collared in significant thicknesses of OB. As a result varying amounts of OB were placed in front of the free face. Free face coverage was complete for L-G #1, severe for L-G #2, and moderate for L-G #4. It was not deemed practical to doze the OB away from the free face prior to the shot. The photograph in Figure 26 shows an expanse of rock face prior to the development of any drill ramps.

2.3.5 L-G #1 Blast Report

Introduction

L-G #1 was the first blast cast in Lake Pit. It was located at the extreme south end of the pit in Cut 2 of the dragline mining sequence.



The photo shows the nature of the blast cast problem in Lake Pit. Firstly, note the 5 - 8m thick layer of OB consisting mainly of ablation glacial till. Below this is the rock unit which exists on top of M Seam. The line of stakes mid way up the picture are located on the crest of the M seam footwall berm. The parting between M seam and Mu seam, a 0.6m seam is variable ranging from 0 to 10m. Where this parting exceeds 2m, Mu is blasted. Where the parting is less than 2m the hole is loaded to top of Mu. The rock above M seam is not massive. Towards the north end of Lake Pit there are several coaly stringers which exist above Mu. The photo shows the face expanse from the north end of L-G #1 to the south end of L-G #4 prior to any ramp development.

Figure 26 Lake Pit Blast Casting - L-G Patterns

General Information

Figure 27 shows a portion of the blast cast in plan view as well as two cross sections.

Drilling was done off of 3 ramps dozed into overburden material as shown in the top photograph in Figure 28. In this instance the variance between actual hole locations and design hole locations was severe. A great amount of reliance was placed on pit technicians to ensure holes were field fit in an acceptable manner.

L-G #1 was not photographed with a high speed camera although a video tape record of this blast was attained.

Pattern Design Information

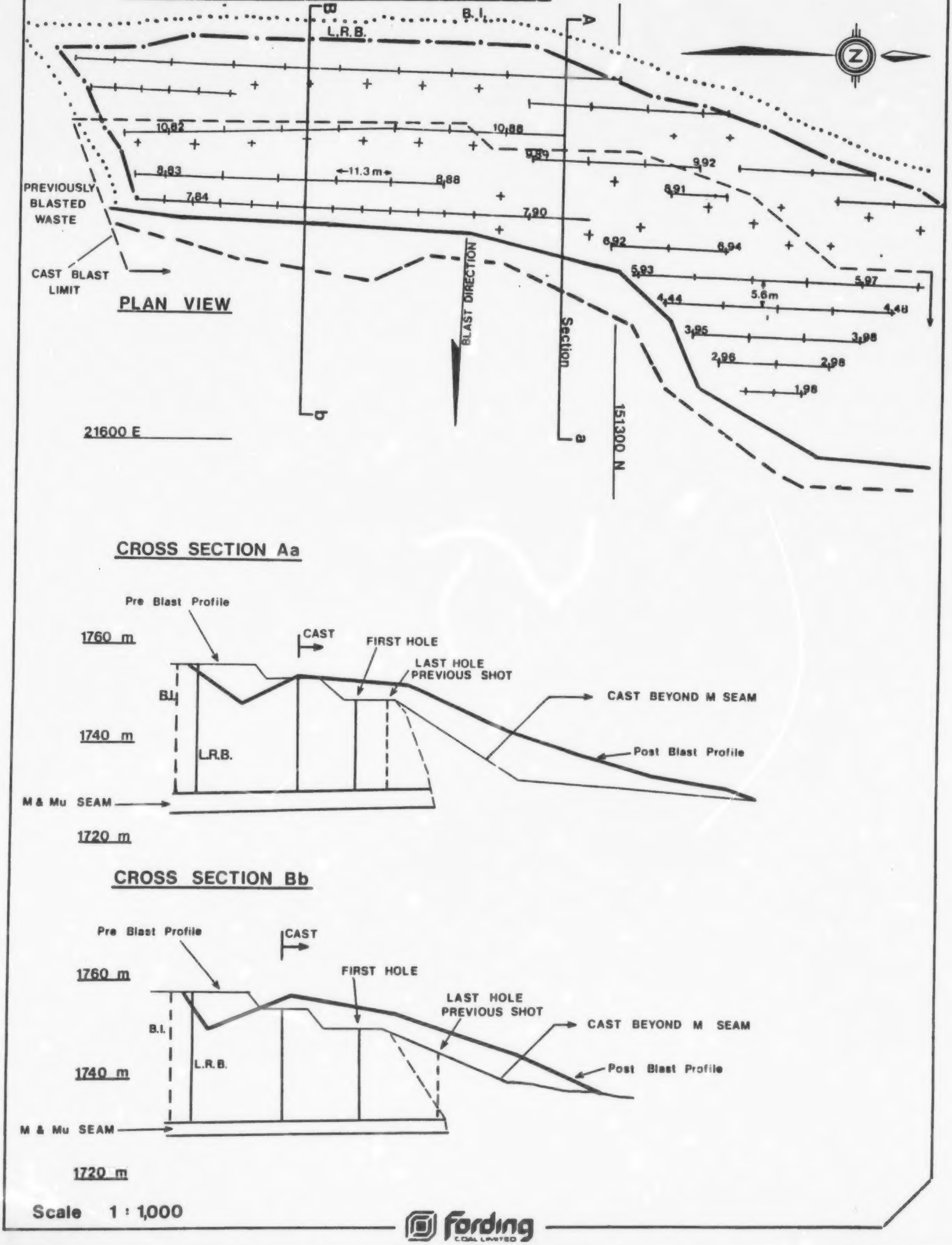
Drilling

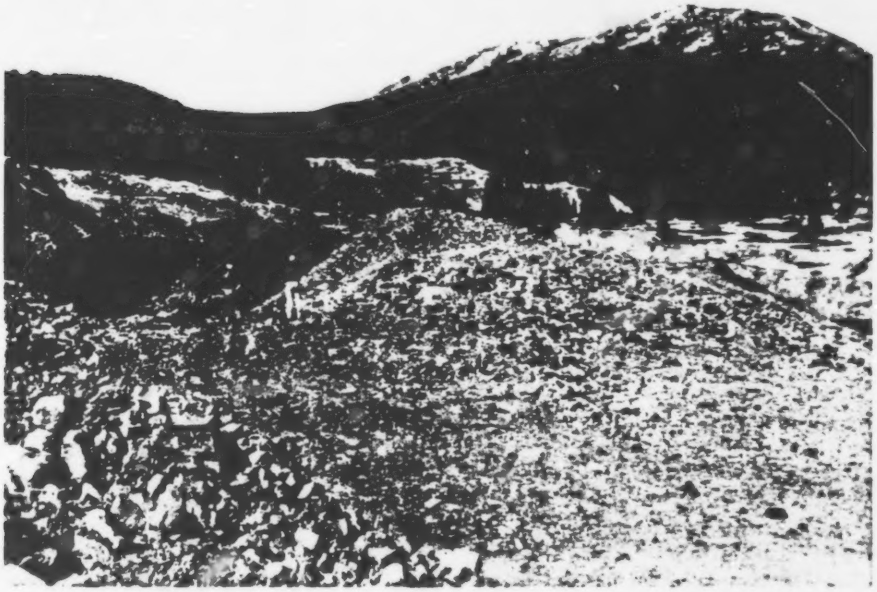
- Hole Diameter - 270mm
- Pattern Dimensions - 5.6 x 11.3; B/S = 0.5
- Angle Drilling Details - N/A
- Average D/W = 0.53; Average L/W = 8.3
- Half space front row holes - only where necessary for excessive toe
- Adverse LRB orientation - No
- Average Front Row Toe Burden - 11m (rock). There was an additional 7 - 25m of OB and blasted rock against toe of the free face.
- General - Ramps had to be cut to make the drill platform. Narrow Ramps particularly in south two-thirds of pattern resulted in severe variance between design and actual collar locations. OB pushed over face during ramp construction placed additional effective burden on the front row of holes.

Loading Details

- Design Loads and Collars - Stand off - front row 1.22m, other rows 0.9m. All holes loaded to 5.8m collars with SuperAN 2510/Mag 9013.
- Actual PF/EF - 0.81 kg/BCM; 90 weu/BCM.

Figure 27 LG 1 Blast Cast


B
B.I.
A
L.R.B.
Z
10.82
10.88
9.89
9.92
8.83
11.3 m
8.88
8.91
7.84
7.90
6.92
6.94
5.93
5.97
5.6m
4.44
4.48
3.95
3.98
2.96
2.98
1.98
PREVIOUSLY BLASTED WASTE
CAST BLAST LIMIT
BLAST DIRECTION
Section
b
a
PLAN VIEW
21600 E
151300 N
CROSS SECTION Aa
Pre Blast Profile
1760 m
CAST
FIRST HOLE
LAST HOLE PREVIOUS SHOT
CAST BEYOND M SEAM
B.I.
1740 m
L.R.B.
Post Blast Profile
M & Mu SEAM
1720 m
CROSS SECTION Bb
Pre Blast Profile
CAST
1760 m
FIRST HOLE
LAST HOLE PREVIOUS SHOT
CAST BEYOND M SEAM
B.I.
1740 m
L.R.B.
Post Blast Profile
M & Mu SEAM
1720 m
Scale 1 : 1,000



The top photo shows completed ramps staked and ready for drilling. Note the extent of free face coverage with OB.

The bottom photo shows a 3 - 4m high power trough along the back of the blast indicating that good forward movement was achieved.

Figure 28 L-G #1 General Photographs

- Percentage Heavy ANFO Explosive - 27%
- Location of Wet Hole Explosive - towards back of blast and towards north end of blast.

Initiation

- DTH delays - period 10 Nonels 2 per hole
- surface delays - 100ms
- blast direction - perpendicular to the free face.

Geology

- Interbedded sandstones, siltstones, and coaly stringers above. M seam. Structure dipping slightly to north. Blast direction due east.

General Comments

- Although toe burden was acceptable, the free face was totally filled in with overburden during the course of ramp construction.

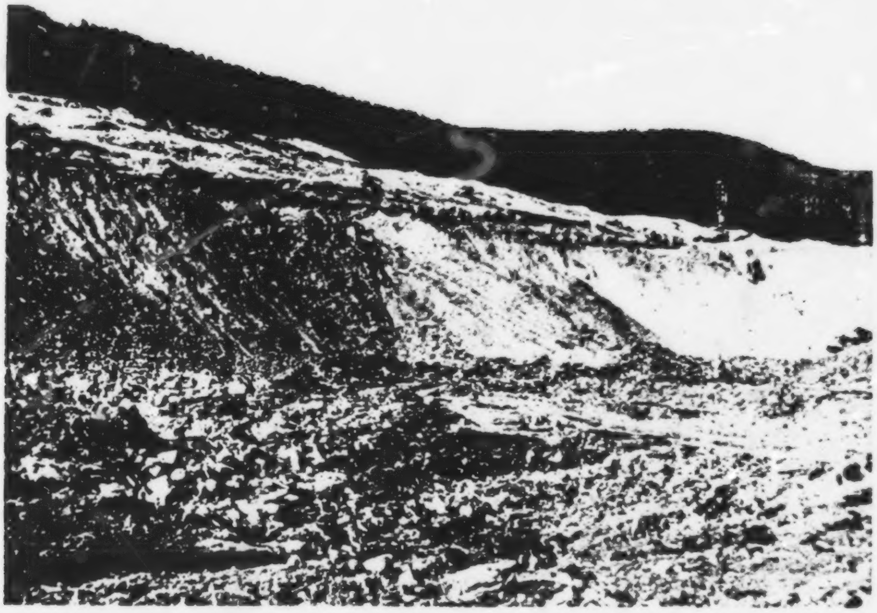
Analysis

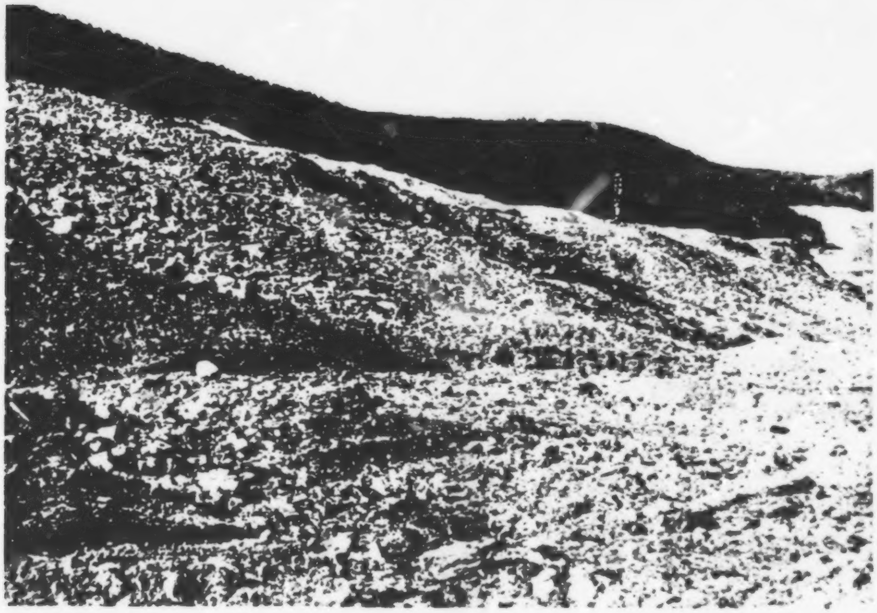
Relevant analysis information was taken off of 3 cross sections, two of which are shown in Figure 27. The photographs in Figure 29 were taken before and after the blast from roughly the same location.

Figure 30 summarizes results from the sectional analysis. A total of 4,400 BCM Rock and 15,300 cm of OB was cast beyond the M seam mined limit. This result was achieved with a PF = 0.81 kg/BCM; 90 weu/BCM.

Cast results are poor probably because of the thick layer of OB restraining movement from the free face. Furthermore, the large number of off pattern holes resulted in a less than optimal explosive distribution. Toe burdens were acceptable for the majority of the front row holes.

Figure 31 summarizes the incremental costs and benefits for L-G #1 in the general format.





Views looking north. Note the extent of the overburden pushed over the free face. For reference note the light colored log in the lower centre of both pictures. Good forward movement was achieved but cast results were poor.

Figure 29 L-G #1 Pre and Post Blast.

Figure 30 L-G #1 Blast Results

Section	D/W	Cast Area BCM/M (rock) cm/m (OB)	% Cast Beyond M Seam
1[*]	.38	570 + 51 - 730.5/S (OB)	9 (s = 1.29)
2	.8	99 (OB) + 84 (rock)	31
3	.7	67 (OB)	9

Total Volume, Vt = 119,500 BCM rock + 22,500 cm OB/1.25 = 137,500 BCM

Volume Cast, Vc = 4,400 BCM rock + 15,300 cm OB/1.25 = 16,640 BCM

Percent Cast = 100 * %Vc = 12.1

Figure 31 L-G #1 Incremental Costs and Benefits

Incremental Cost

$= 26{,}199\ U_{2510} + 71{,}254\ U_{9013} - 15{,}194\ U_{2495} - 56{,}490\ U_{9000}$ } explosives

$+ 42\ (U_P + U_N)$ } accessories

$+ 442\ U_D$ } drilling

Incremental Benefits

$= (Vc - Vpc)\ U_{DLM}$

$= (Vc - Vt * \%\ Vpc)\ U_{DLM}$

$= (16{,}640 - 137{,}500 * \%\ Vpc)\ U_{DLM}$

For % Vpc = 0.05 estimated

$= 9{,}765\ U_{DLM}$

---

[*] Due to an incomplete survey of post blast topography. the analysis of cast volumes for this section only was based on the "open" section technique. For sections 2 and 3 an average swell factor of 1.29 was back calculated and used for the cast evaluation in section 1.

2.3.6 L-G #2 Blast Report

Introduction

The L-G #2 pattern is located immediately north of L-G #1. In general, hole depths are greater in L-G #2 compared to L-G #1 providing a more beneficial blast casting geometry.

Design Information

Figure 32 shows a plan and a representative cross section of L-G #2. The photograph in Figure 33 shows ramp construction underway with a dozer pushing OB off the face onto the M seam footwall berm.

Pattern Design Information

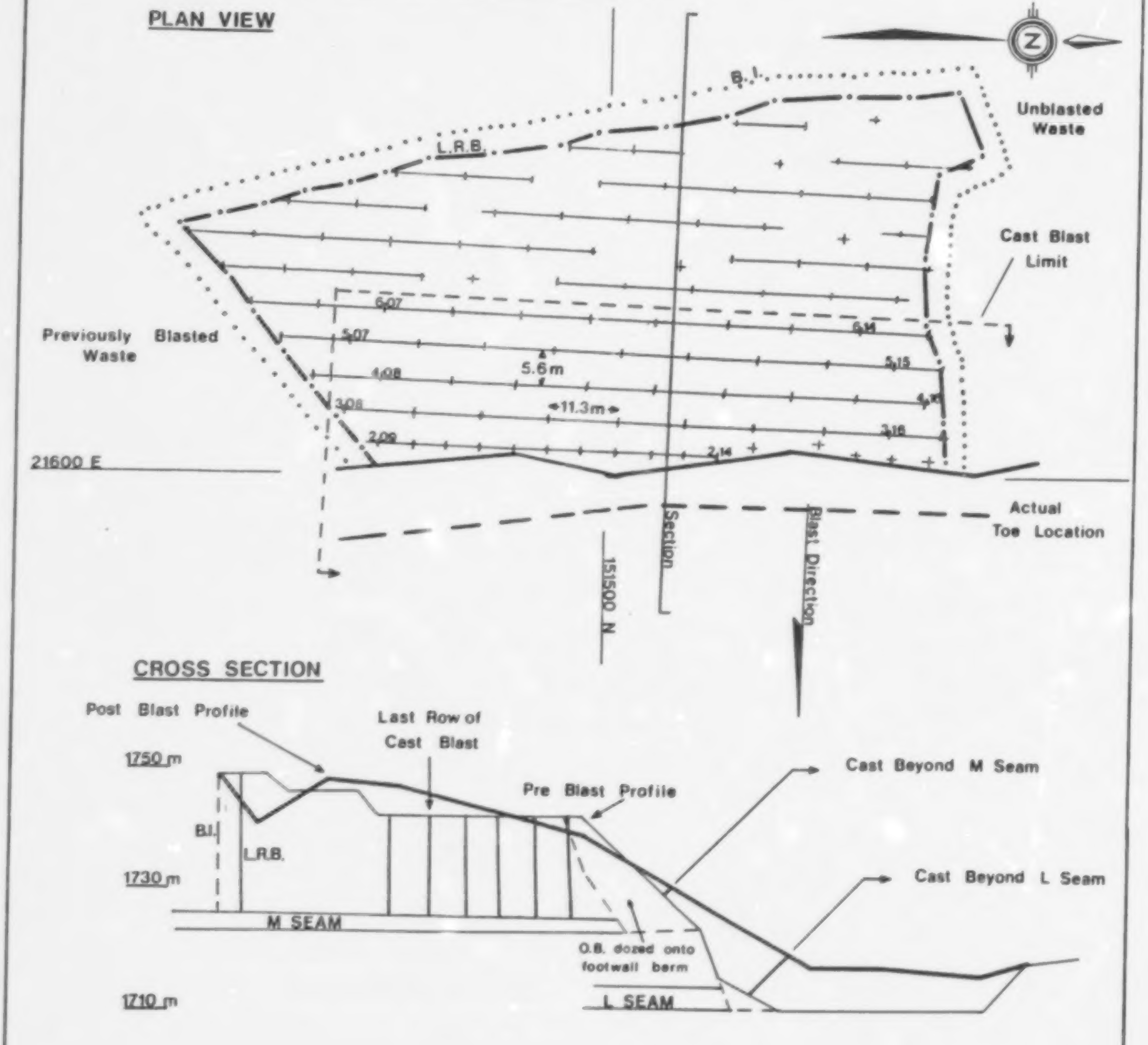
Drilling

- Hole Diameter - 270mm
- Pattern Dimensions - 5.6 x 11.3 staggered; B/S = 0.5
- Angle Drilling Details - N/A
- Average D/W =0.63 to rock contact; 0.49 to OB surface; Average L/W = 3.0
- Average Front Row Toe Burden - 9m to rock contact; 19m to OB contact.
- Adverse LRB orientation - south end pattern bounded by broken muck from L-G #1. North end LRB is parallel to blast direction which has adverse casting effects.
- General - although ramps had to be constructed almost all blast cast holes were located on one large ramp. The drill could nose out to get crest holes and deviation between actual collar locations and design locations was minimal. Crest holes half spaced.

Loading Details

- Design Loads and Collars - Stand off - crest holes 1.2m, remaining holes 0.9m. All holes loaded to 5.8m collars with SuperAN 2510/

Figure 32 LG 2 Blast Cast



PLAN VIEW

CROSS SECTION

Z
B.I.
L.R.B.
Unblasted Waste
Cast Blast Limit
Previously Blasted Waste
6.07
5.07
4.08
3.08
2.09
6.14
5.15
4.16
3.16
2.14
5.6m
11.3m
21600 E
Actual Toe Location
Section
Blast Direction
151500 N
Post Blast Profile
Last Row of Cast Blast
1750 m
1730 m
1710 m
B.I.
L.R.B.
Pre Blast Profile
Cast Beyond M Seam
Cast Beyond L Seam
M SEAM
L SEAM
O.B. dozed onto footwall berm

Scale 1 : 1,000

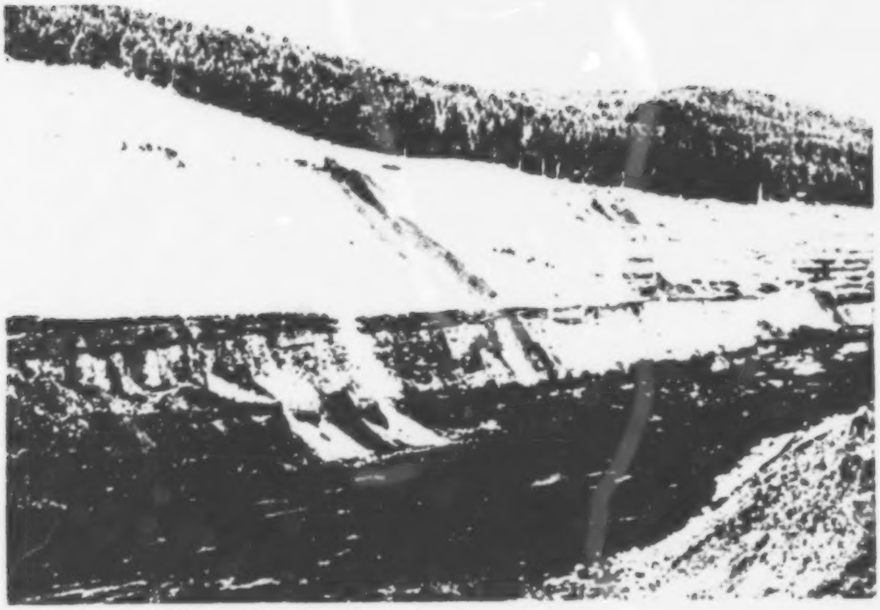


Photo shows a dozer pushing the OB off of the drill platform. At this stage, near completion of ramp construction, the footwall berm is almost completely full.

The face to the right (north) of the dozer push is the location of L-G #4.

The interbedded nature of the rock above M seam is clearly visible.

Figure 33 L-G #2 Ramp Construction

Magnafrac 9013

- Actual PF/EF - 0.97 kg/BCM / 89 weu/BCM
- Percentage Heavy ANFO Explosive - 88%.
- Location of Wet Hole Explosive - back rows towards north end.

Initiation

- DTH delays - period 10 Nonels 2 per hole
- surface delays - 100ms delay
- blast direction - perpendicular to free face, slightly up dip towards 090°.

Geology

- structure dipping at 3° towards 210°.
- Hanging wall rock on M seam comprised of interbedded sandstone, siltstone, and mudstone with occasional coaly stringers (See figure 26).

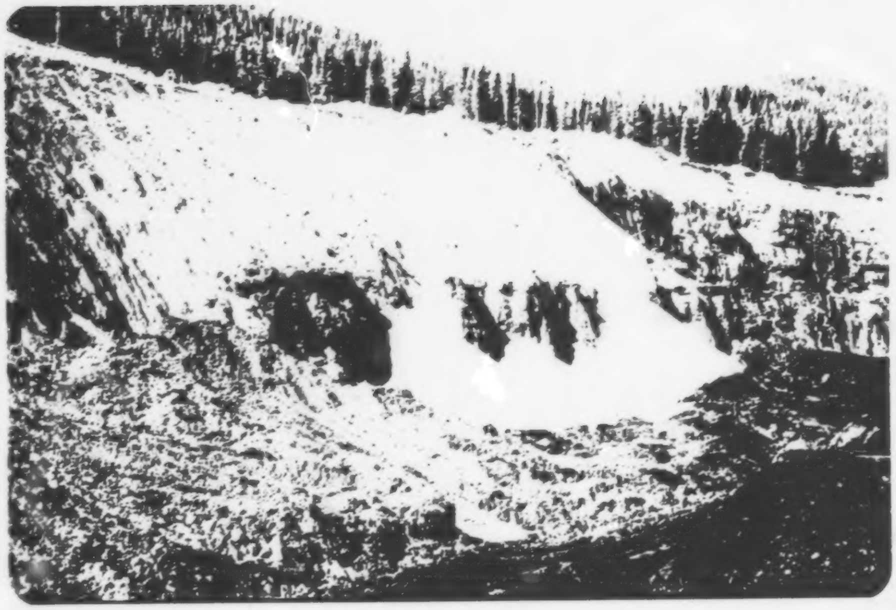
Analysis

Figure 34 shows photographs taken before and after the shot. Targets were placed on the face and near the collars of holes in order to determine face and collar velocities. As soon as the OB on the face began to move the targets were completely obscured so that velocity information could not be attained. It is apparent that casted volumes in L-G #2 were much higher than for L-G #1.

Five (5) sections were evaluated in order to determine the appropriate cast volume for L-G #2. The results are summarized in Figure 35.

42,700 BCM of Rock was drilled and loaded for casting. Approximately 9,600cm of OB was located in front of the free face and subject to casting as well. It is estimated that 17,500 BCM of rock and all of the OB

Photo taken prior to shot. Note the full coverage of the free face with OB. Also note the collar and face targets located for high speed photography analysis.



Post Blast photo of L-G #2.

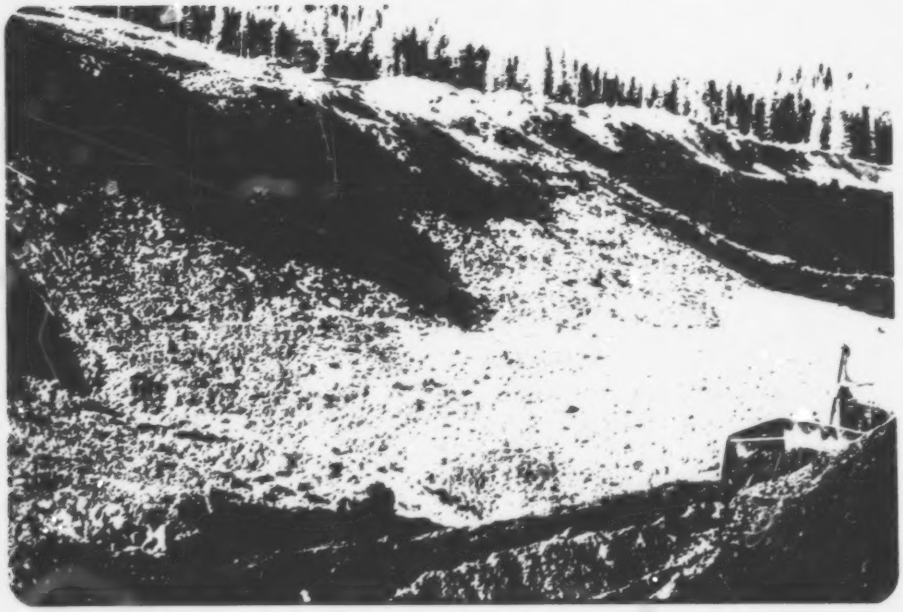


Figure 34 L-G #2 Pre and Post Blast Shots

Figure 35 L-G #2 Blast Results

Section	D/W	Area Cast Beyond M Seam LMS	Area Cast Beyond L Seam LMS	Pre Blast Area		Cast Beyond M Seam	% Cast Beyond L Seam
				Rock BSM	Over Burden LSM		
1	0.46	302	210	425	113	46	32
2	0.66	387	267	417	114	59	41
3	0.55	332	236	401	91	54	39
4	0.54	355	248	420	72	57	40
5	0.50	271	186	500	67	38	26

Total Volume, Vt = 42,678 BCM rock + 9,636 CM OB/1.25
= 50,169 BCM
Volume Cast, Vc_M beyond M Seam = 17,527 BCM rock + 9,363 CM OB/1.25
= 25,017 BCM
Percent Cast (100 * $\%Vc_M$) beyond M Seam = 50

Volume Cast, Vc_L beyond L Seam = 11,530 BCM rock + 8,205 CM OB/1.25
= 18,094 BCM
Percent Cast (100 * $\%Vc_L$) beyond L Seam = 36

material was cast beyond M seam and 11,500 BCM rock and 8,200 cm of OB was cast beyond L seam. These results were achieved with an EF = 89 weu/BCM. The comparable production blast would have used approximately 56 weu/BCM.

The overburden on the free face was recognized as being detrimental for blast casting but it was not practical to remove it. Crest holes were half spaced in order to assist in overcoming the resistance to forward movement mobilized by the OB. Regular spacing would have been sufficient to overcome the front row toe burden for a production shot.

The incremental costs associated with L-G #2 resulted from drilling and loading extra holes resulting from the pattern reduction and the half spacing of crest holes. Furthermore, higher bulk strength explosives were used than for standard production. Collars were raised 0.9m over production collars. The general expression for incremental costs and benefits is shown in Figure 36.

2.3.7 L-G #4a and #4b Blast Report

Introduction

L-G #4 blast cast was located immediately north of L-G #2. L-G #4 was divided into 2 distinct shots which were shot simultaneously L-G #4a (south side) and L-G #4b (north side).

Pattern Design Information

Relevant pattern design information for L-G #4a and L-G #4b are itemized in Figures 37 and 38 respectively.

The phototgraphs in Figure 39 illustrate the difficulties associated with drilling on ramps. Difficulties arise in getting holes located on pattern and trying to achieve a reasonable toe burden on the front row. A photograph in Figure 40 shows the completed drill pattern and the entire ramp system that was employed.

Figure 36 L-G #2 Incremental Costs and Benefits

Incremental Cost

$= 36,561U_{2510} + 5,122U_{9013} - 15,766U_{2495} - 5,905U_{2505}$ }

} explosives

$- 4,103U_{9000}$ }

$+ 30\ (U_P + U_N)$ } accessories

$+ 302\ U_D$ } drilling

Incremental Benefits for L and M Seams respectively

$= [(Vc_L - Vpc_L)U_{DLL} + (Vc_{ML} - Vpc_{ML})]U_{DLM}$

$= (Vc_L - 0.7 * Vt * \%Vpc)U_{DLL} + (Vc_{ML} - 0.3 * Vt * \%\ Vpc)U_{DLM}$

Where 0.7, 0.3 are factors to proportion the benefit of the production cast volume relative to Vt between L and M seams. Vc_{ML} and Vpc_{ML} are defined as volumes cast beyond M seam but in front of L seam.

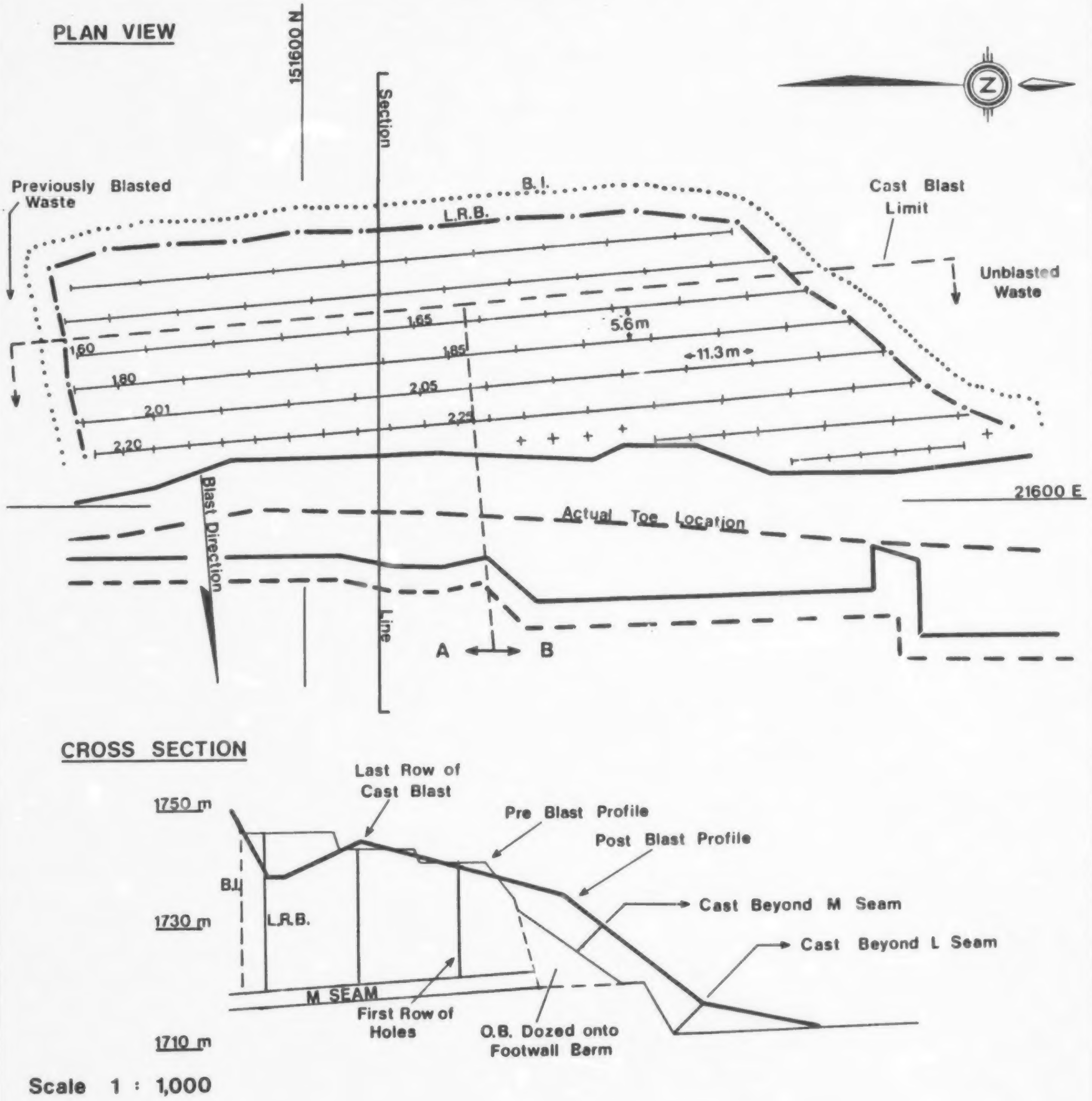
$= (18,094 - 0.7 * 50,168 * \%Vpc_L)U_{DLL} + (6,923 - 0.3 * 50,168 * \%Vpc)U_{DLM}$

$\%Vpc_L = 0.7 * Vpc;\ \%Vpc_{ML} = 0.3 * \%Vpc$

For $\%Vpc = 0.10$

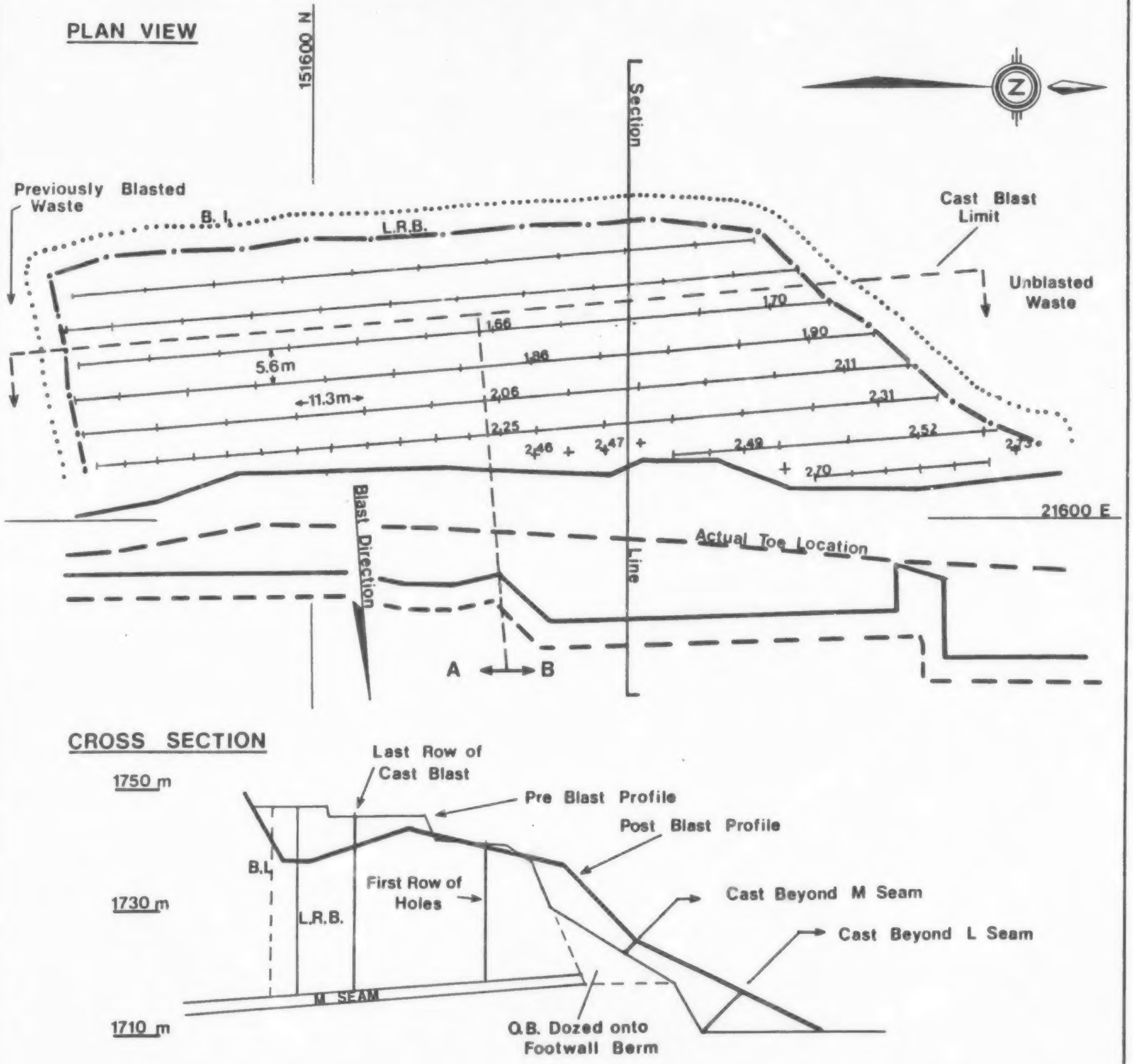
$= 14,582\ U_{DLL} + 5,418\ U_{DLM}$

Figure 37 LG 4A Blast Cast


PLAN VIEW
151600 N
Section Line
Previously Blasted Waste
B.I.
L.R.B.
Cast Blast Limit
Unblasted Waste
1.65
5.6m
1.60
1.85
11.3m
1.80
2.05
2.01
2.25
2.20
21600 E
Blast Direction
Actual Toe Location
A
B
CROSS SECTION
Last Row of Cast Blast
1750 m
Pre Blast Profile
Post Blast Profile
B.I.
1730 m
L.R.B.
Cast Beyond M Seam
Cast Beyond L Seam
M SEAM
First Row of Holes
O.B. Dozed onto Footwall Berm
1710 m
Scale 1 : 1,000

Fording Coal Limited

Figure 38 LG 4B Blast Cast


PLAN VIEW
151600 N
Section Line
Z
Previously Blasted Waste
B.L
L.R.B.
Cast Blast Limit
Unblasted Waste
1.70
1.66
1.90
5.6m
1.86
2.11
11.3m
2.06
2.31
2.25
2.52
2.46
2.47
2.49
2.73
2.70
21600 E
Blast Direction
Actual Toe Location
A
B
CROSS SECTION
1750 m
1730 m
1710 m
Last Row of Cast Blast
Pre Blast Profile
Post Blast Profile
B.L
L.R.B.
First Row of Holes
Cast Beyond M Seam
Cast Beyond L Seam
M SEAM
O.B. Dozed onto Footwall Berm

Scale 1 : 1,000

This photo shows the difficulties in maintaining a reasonable toe burden when drilling crest holes on a narrow ramp. The inability of the drill to nose out to the crest results in additional front row burdens. Drill stability is also of concern. Also note the extent of which the free face is covered with OB.

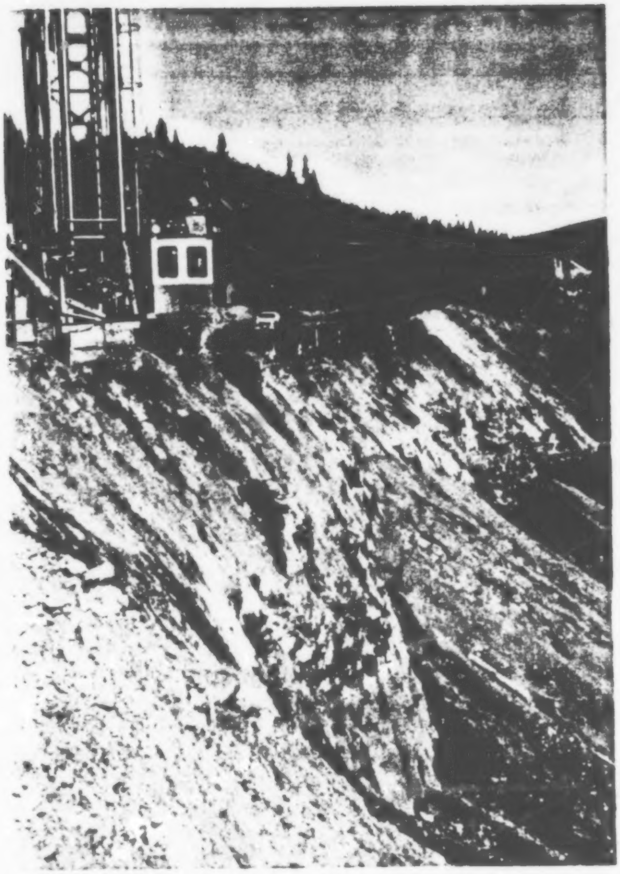


Photo shows the front 4 rows of the blast. Limited drill mobility coupled with narrow ramps results in off pattern holes. Note extreme difficulties which arise if the drill has to go back for redrills.

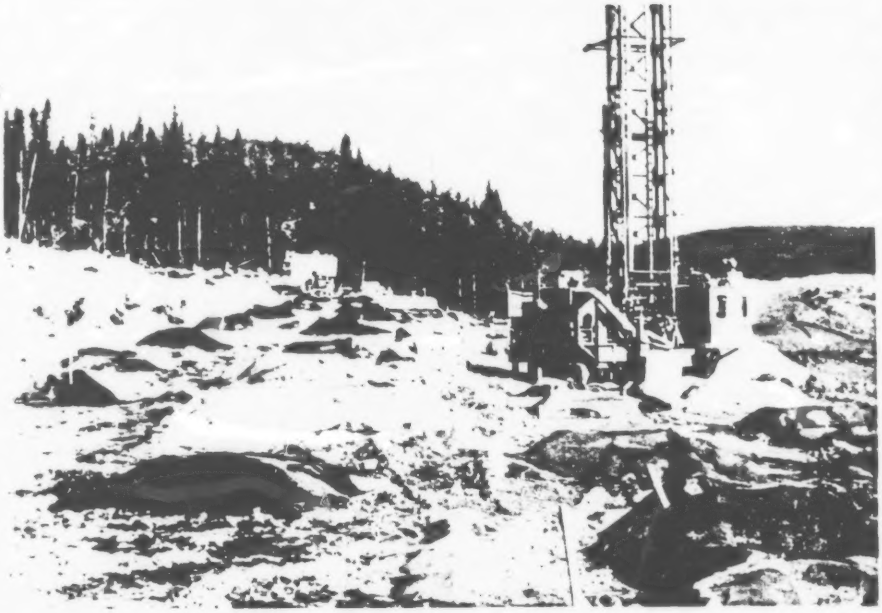


Figure 39 L-G #4 Drilling Narrow Ramps.

Drill pattern ready to shoot. Collar and face targets in place for high speed film analysis. Note the OB wedge on top of the M seam footwall berm. The pattern at the far end of the water was not a part of the blast cast.

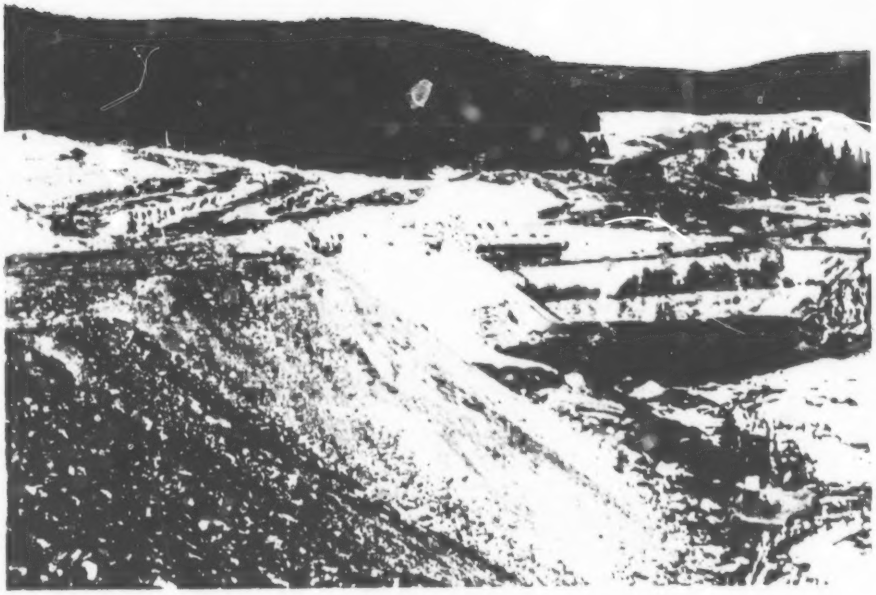


Photo shows muck pile profile after the shot.

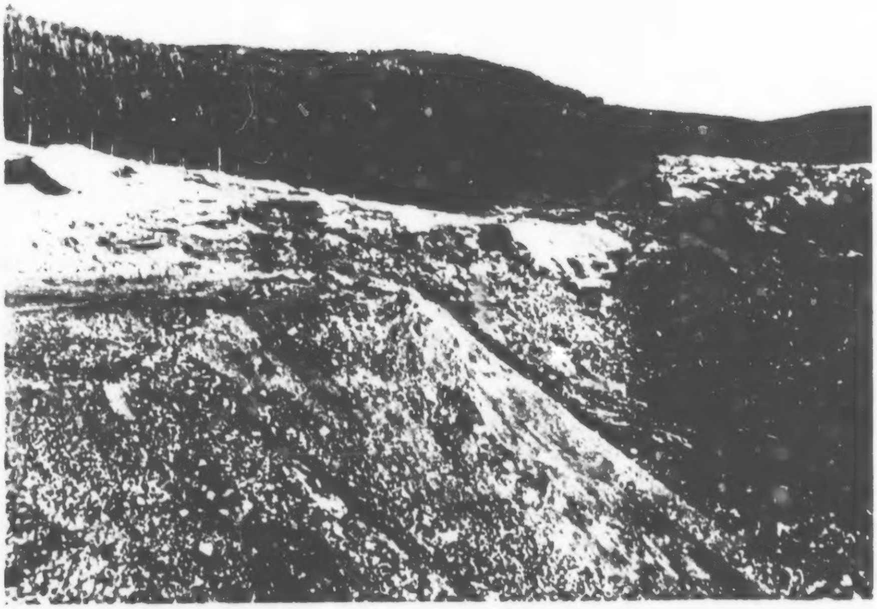


Figure 40. L-G #4 Pre and Post Blast Photographs

Pattern Design Information (L-G #4a)

Drilling

- Hole Diameter - 270 mm
- Pattern Dimensions - 5.6 x 11.3 staggered; B/S = 0.5
- Angle Drilling Details - N/A
- Average D/W = .68; Average L/W = 2.6 (not bounded by unblasted rock)
- Average Front Row Toe Burden = 13m
- Adverse LRB orientation - none
- General - crest holes half spaced to account for large toe burdens all holes drilled to coal.

Loading Details

- Design Loads and Collars - stand off 1.2m for crest holes, 0.91m for subsequent rows; collars 6.4m, loaded with SuperAN 2510/Mag9013 all holes.
- Actual PF/EF - .91 kg/BCM / 82 weu/BCM
- Percentage of Heavy ANFO Explosive - 0%
- Location of Wet Hole Explosive - entire pattern

Initiation

- DTH delays - period 10 Nonels 2 per hole
- surface delays - 150ms
- blast direction - perpendicular to free face. Blast direction slightly up dip towards 084°.

Geology

- structure dipping at 6 - 10° towards 319°.
- interbedded sandstones and siltstones. 2 - 3 coaly stringers in face at varying distances above Mu seam.

Pattern Design Information (L-G #4b)

Drilling

- Hole Diameter - 270mm
- Pattern Dimensions - 5.6 x 11.3 staggered; B/S =0.5
- Angle Drilling Details - N/A
- Average D/W = .77; Average L/W = 2.2 (bounded by unbroken waste on north side only
- Average Front Row Toe Burden - 16m
- Adverse LRB orientation - none
- General - crest holes half spaced to account for large toe burdens. All holes drilled to coal.

Loading Details

- Design Loads and Collars - same standoff as L-G #4a. Collars 6.4m. Crest holes loaded with SuperAN 2510/Mag9013. Subsequent rows - 50% column loaded with SuperAN 2510/Mag9013. Top 50% of column loaded with SuperAN 2495/Mag9000.
- Actual PF/EF - .99 kg/BCM / 86 weu/BCM
- Percentage Heavy ANFO Explosive - 0%
- Location of Wet Hole Explosive - entire pattern

Initiation

- DTH delays - period 10 Nonels 2 per hole
- surface delays - 150ms
- blast direction - perpendicular to free face.

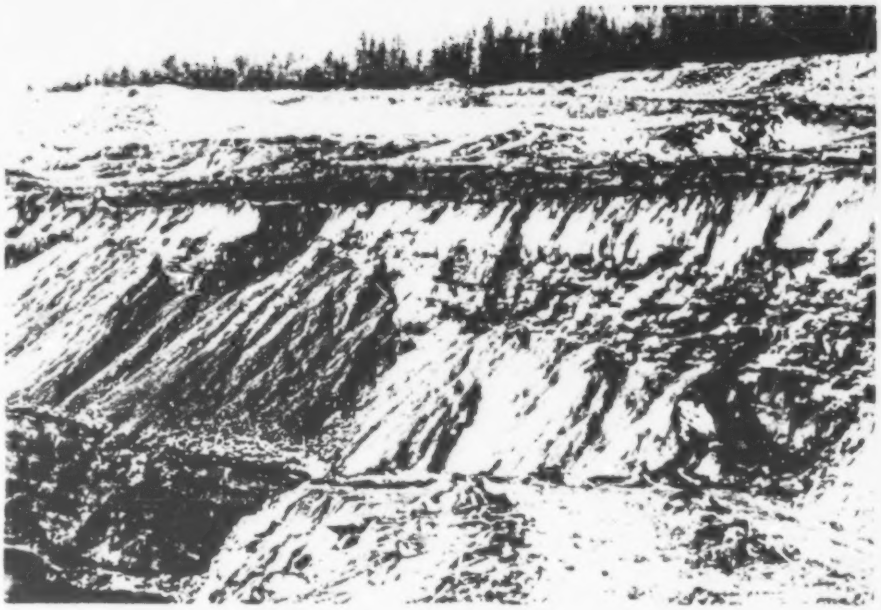
Geology

- same as L-G #4a.

Analysis

Figures 40 and 41 contain photographs showing pre and post blast information. The nature of the free face and the amount of overburden against the free face is clearly shown. Several targets were suspended on the free face against the OB surface and against the hard rock face in

Photo illustrates free face. Note truck at top right corner and personnel walking along crest for scale. M seam footwall berm completely filled with OB at south end but only partially filled at north end. Holes extend approximately down to top of footwall berm. Pattern in foreground is not a blast cast.



Post Blast picture from vantage point as above. Note personnel standing on back of power trough immediately below the dozer blade in the background.

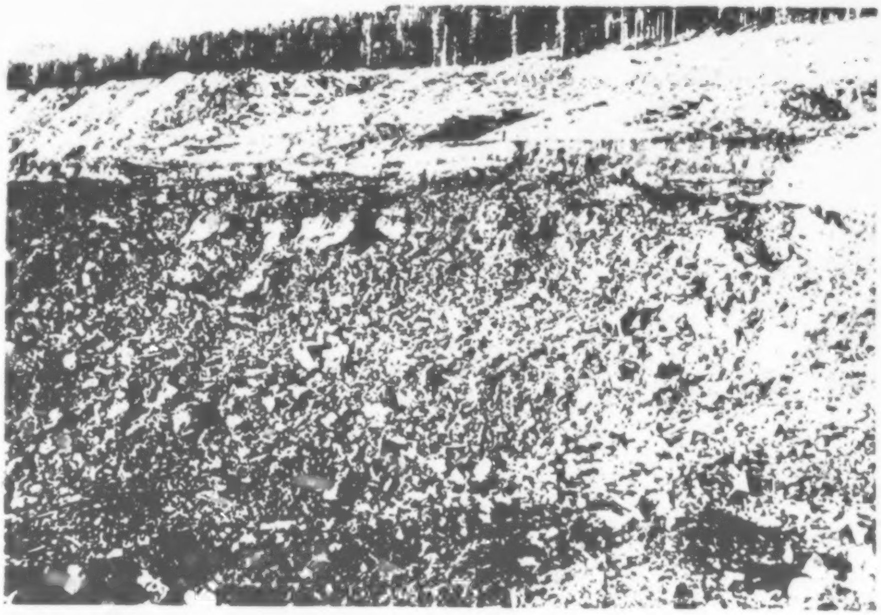


Figure 41. L-G #4 Pre and Post Blast Photographs

order to attain face velocity information at these points. Targets were also located adjacent to hole collars so that collar velocity information could be attained. Unfortunately the film turned out poorly and the necessary velocity information could not be attained.

Necessary analysis information was obtained by evaluating six (6) cross sections with pre and post blast information. A summary of this information for both blasts is located in Figure 42.

L-G #4a

The analysis shows that with an EF = 82 weu/BCM and a PF = .91 kg/BCM and with an average D/W ratio = 0.68, 23% of the waste rock blast casted was cast beyond M seam, 10% beyond L seam. Furthermore, it is estimated that 100% of the OB was cast beyond M seam and 32% beyond L seam. The average toe burden on the front row of holes being approximately 13m.

L-G #4b

The analysis shows that with an EF = 86 weu/BCM and a PF = .99 kg/BCM, and with an average D/W = 0.77, 21% of the rock was cast beyond M seam, 9.4% beyond L seam. Additionally it is estimated that 100% of the OB was cast beyond M seam and 47% beyond L seam. The average front row toe burden was 16m for this area of the pattern.

Incremental costs associated with L-G #4a and L-G #4b are associated with drilling and loading extra holes resulting from the pattern spacing reduction. The half spacing of holes on the front row would have been necessary for a standard blast because of the excessive toe burden along the front row. For the blast cast all crest holes were loaded to the same collar, however, for a production blast alternate, crest holes would have been toe loaded only. Incremental costs are also accrued due to the use of higher bulk strength explosives selected for casting and the reduced collar compared to a standard production shot.

Figure 43 summarizes the incremental costs and benefits in general terms.

Figure 42 L-G #4 Blast Results

Section	D/W	Area Cast Beyond M Seam LSM	Area Cast Beyond L Seam LSM	Pre Blast Area		% Cast Beyond M Seam	% Cast Beyond L Seam	Blast
				Rock BSM	Over Burden LSM			
0	0.85	271	144	560	22	36	19	4b
1	0.71	153	82	768	57	16	9	
2	0.73	151	64	728	53	16	7	
3	0.76	156	54	661	62	19	6	
4	0.70	178	60	537	51	25	9	4a
5	0.67	202	100	487	38	31	14	

4a Total Volume, Vt = 35,300 BCM rock + 4,000 CM OB/1.25
= 38,500 BCM

Volume Cast, Vc_M beyond M Seam = 8,100 BCM rock + 4,000 CM OB/1.25
= 11,300 BCM

Percent Cast (100 * %Vc_M) beyond M Seam = 29.4

Volume Cast, Vc_L beyond L Seam = 3,600 BCM rock + 1,300 CM OB/1.25
= 4,640 BCM

Percent Cast (100 * %Vc_L) beyond L Seam = 12.0

4b Total Volume, Vt = 60,614 BCM rock + 3,920 CM OB/1.25
= 63,750 BCM

Volume Cast, Vc_M beyond M Seam = 12,500 BCM rock + 3,920 CM OB
= 15,636 BCM

Percent Cast (100 * %Vc_M) beyond M Seam = 24.5

Volume Cast, Vc_L beyond L Seam = 5,700 BCM rock + 1,840 CM OB
= 7,172 BCM

Percent Cast (100 * Vc_L) beyond L Seam - 11.3

NOTES: Assume 100% of mud wedge cast beyond M seam. Proportion of OB cast beyond L is same as proprotion of total area beyond seams. Volumes to be casted included the overburden wedge.

Sample calculation % cast beyond M seam Section 0

$$= \frac{(271 - 22)/1.3 + 22/1.25}{560 + 22 / 1.25} = 0.36$$

Figure 43 L-G #4 Incremental Costs and Benefits

L-G #4a	L-G #4b
Incremental Costs $= 25,222\ U_{9013} - 20,678\ U_{9000}$ $+ 6\ (U_P + U_N)$ $+ 55\ D$	Incremental Costs $= 28,304\ U_{9013} - 16,299\ U_{9000}$ $+ 8\ (U_N + U_P)$ $+ 113\ U_D$
Incremental Benefits for L & M Seam $= (Vc_L - Vpc_L)U_{DLL} + (Vc_{ML} - Vpc_{ML})U_{DLM}$ $= (Vc_L - 0.4 * Vt * \%Vpc)U_{DLL}$ $+ (Vc_{ML} - 0.6 * Vt * \%Vpc)U_{DLM}$ where 0.4, 0.6 are factors to proportion the benefit of the production cast volume relative to Vt Define $Vpc_L = 0.4 * Vpc$; $Vpc_{ML} = 0.6 * Vpc$ $= (4,640 - 0.4 * 38,500 * \%Vpc)U_{DLL}$ $+ (6,660 - 0.6 * 38,500 * \%Vpc)U_{DLM}$ $= 4,640\ (1 - 3.32 * \%Vpc_L)U_{DLL}$ $+ 6,660\ (1 - 3.47 * \%Vpc_M)U_{DLM}$ For $\%\ Vpc = 0.07$ $= 3,562\ U_{DLL} + 5,043\ U_{DLM}$	Incremental Benefits for L & M Seam $= (Vc_L - 0.5 * Vt * \%Vpc)U_{DLL}$ $+ (Vc_{ML} - 0.5 * Vt * \%Vpc)U_{DLM}$ where 0.5 is a factor .... $= (7,172 - 0.5 * 63,750 * \%Vpc)U_{DLL}$ $+ (8,464 - 0.5 * 63,750 * \%Vpc)U_{DLM}$ $= 7,172\ (1 - 4.44 * \%Vpc)U_{DLL}$ $+ 8,464\ (1 - 3.77 * \%\ Vpc)U_{DLM}$ For $\%Vpc = 0.07$ $= 4,941\ U_{DLM} + 6,232\ U_{DLM}$

2.3.8 LJ #1 BLAST REPORT

Introduction

The LJ #1 pattern is located in the extreme south end of Lake Pit, immediately west of L-G #1.

General Information

Figure 44 shows a plan view and a typical cross section of the LJ pattern. Angle hole drilling was essential for this pattern in order to provide a reasonable toe burden. Depths to coal particularly in the north end of the pattern were in excess of 30m. At these depths even steeply dipping face angles would result in excessive toe burdens if angle drilling was not available.

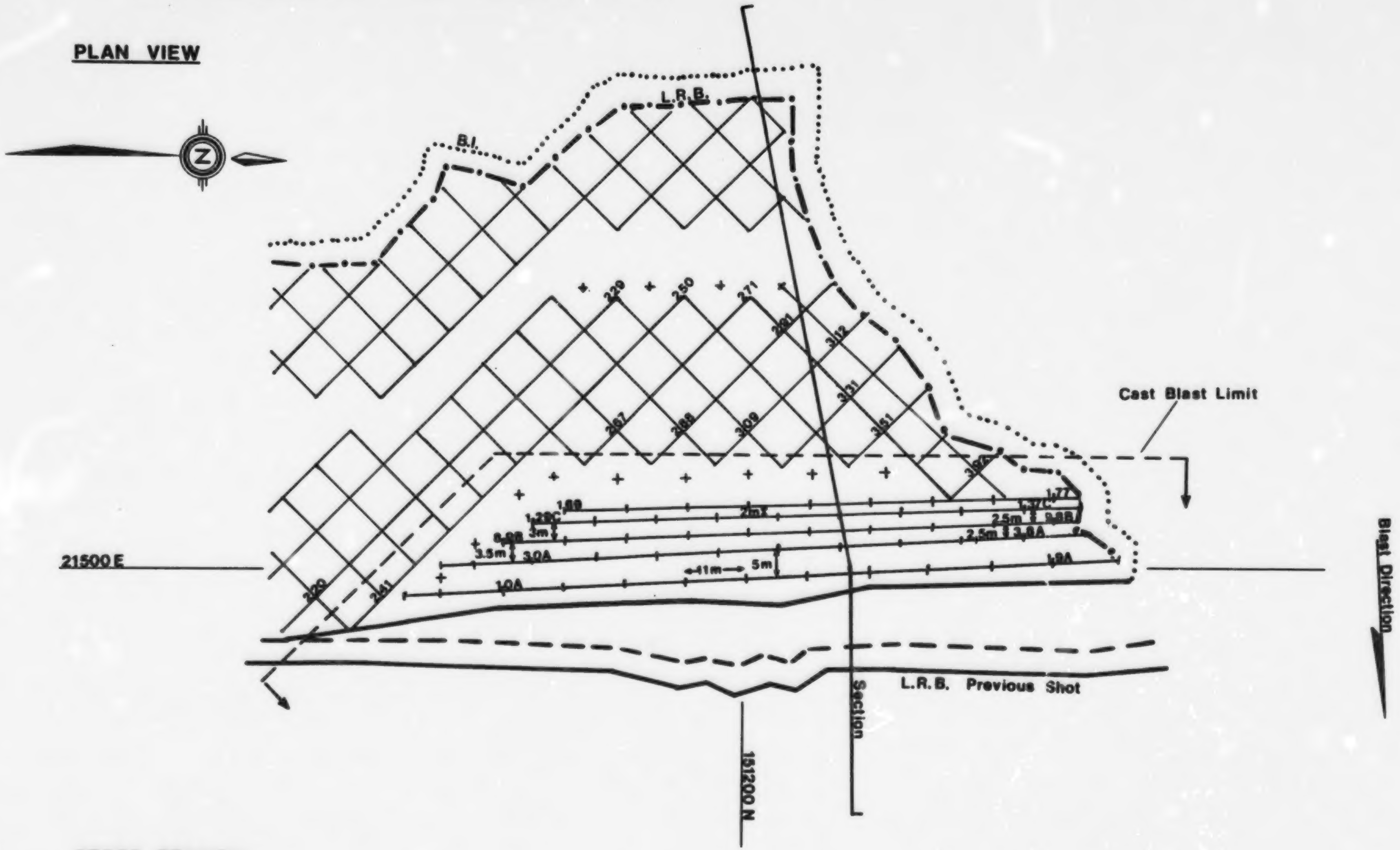
The adjacent LRB to the front row of angle holes is located to the west of the dug out toe limit, indicating that some previously blasted muck had been left in place. Although the actual burden from the bottom of the hole to the rock contact averaged about 12m, the outside segment of this zone was blasted resulting in an assumed average effective burden between 8 and 10m.

The photograph in Figure 45 provides good qualitative information about the nature of the rock strata above M seam. Twelve of the 15 targets suspended on the face are visible in this picture.

The most significant difficulties with the pattern design were that many of the angle holes could not be loaded until most had been drilled because of the close proximity of collars. The resulting time lag resulted in a high exposure to caving holes. Deviation between design and actual hole locations ranged from 0 - 3m in the blast cast region. Off pattern holes were most severe at the crest where redrills were encountered.

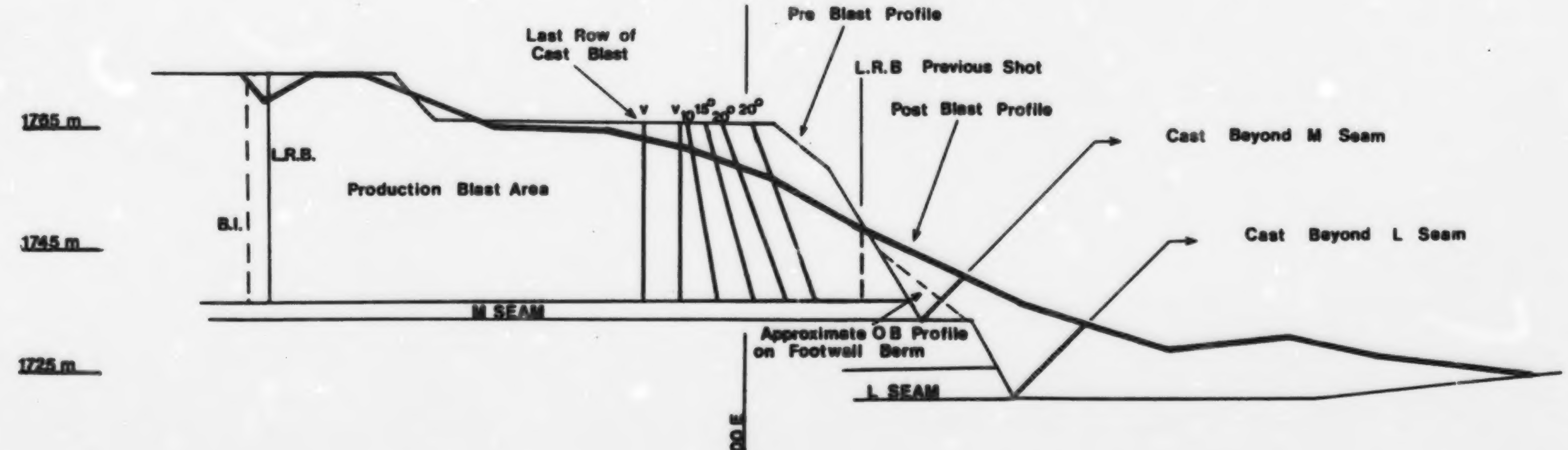
Figure 44 LJ 1 Blast Cast

PLAN VIEW



Z
L.R.B.
B.I.
229
250
271
291
312
331
351
392
267
288
309
220
241
1.89
1.29C
0.9B
3m
3.5m
3.0A
2mX
1.77
1.37C
2.5m
0.8B
2.5m
3.0A
1.9A
11m
5m
1.0A
Cast Blast Limit
21500 E
Blast Direction
L.R.B. Previous Shot
Section
151200 N

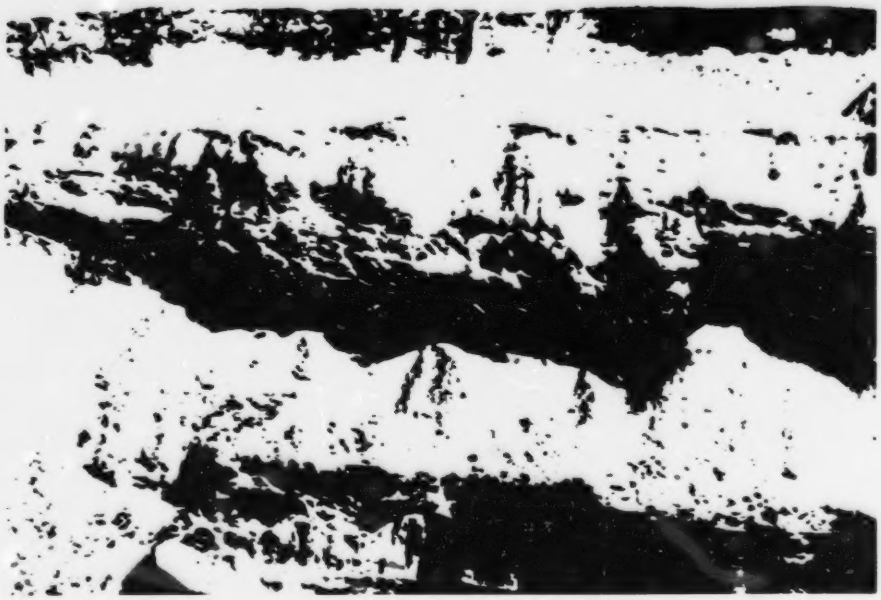
CROSS SECTION



Pre Blast Profile
Last Row of Cast Blast
L.R.B Previous Shot
Post Blast Profile
Cast Beyond M Seam
Cast Beyond L Seam
1765 m
1745 m
1725 m
L.R.B.
B.I.
Production Blast Area
15°
20°
20°
M SEAM
Approximate O B Profile on Footwall Berm
L SEAM
21500 E

Scale 1 : 1,000

Fording



Holes are loaded to the top of Mu or M seam depending on parting thickness, this corresponds to 1 - 2m above the surface elevation of the M seam footwall berm. The bottom 1/4 of the free face is confined moderately by the overburden material dozed onto the footwall berm. Along the south end of the pattern (the left side of the photo) the overburden confinement at the toe increases.

Crest hole depths range from 20m at the south end of the pattern to 33m at the north end. Angle holes were a necessity in this application in order to maintain reasonable front row toe burdens over such a high rock face.

Figure 45 L-J #1 Pre Blast Photograph

Pattern Design Information

Drilling

- Hole Diameter 270mm.
- Pattern dimensions average 5.5x11m staggered B/S = 0.5
- Angle drilling details - front 2 rows at 20^0, 3rd row at 15^0, 4th row at 10^0, 5th and 6th rows vertical.
- Average D/W = 0.76; Average L/W = 4.2
- Average front row toe burden - 12m. The effective burden is less than 12m because the outside portion of this zone had been previously blasted.
- Adverse LRB orientation - none.
- General - when drilling rows of holes at different angles in order to maintain a 5.5m toe burden the collar burdens must be reduced. The offset from row to row was determined in order to maintain constant toe burdens at top of coal. A minimum separation of 2m was needed.

Loading Details

- Design loads and collars - All blast cast holes standoff = 0.9m. Front 2 rows - load to 6.4m collar with SuperAN 2515. Remaining rows - loaded to 6.4m collar with SuperAN 2510
- Actual PF/EF = 0.84 kg/BCM / 76.5 weu/BCM
- Percentage Heavy ANFO - 100%
- Location of wet hole explosive - N/A

Initiation

- DTH delays - period 10 Nonels 2 per hole
- Surface delays - 150ms
- Blast direction perpendicular to free face, along strike

Geology

- Structure dipping 0-5^0 to the north. Interbedded sandstones, siltstones and minor coal stringers above Mu horizon.

General Comments

- The adjacent production blast pattern was 8.5m x 8.5m.
- The free face over the majority of the blast was rock. Towards the south end of the blast cast pattern significant thickness of overburden on the M seam footwall berm were encountered.

Analysis

The cross section illustrated in Figure 44 shows the pre blast and post blast profiles. Figure 46 shows a table of results developed from 7 sections drawn through the blast.

A total of 70,000 BCM's were cast beyond M seam mined limit and 51,200 BCM's were cast beyond L seam mined limit. Sixty-six percent of the volume in the blast cast was cast beyond M seam, 48% was cast beyond L seam. The volume of overburden was not small enough to be separated out in the analysis. The photographs taken before and after the shot in Figure 47 and Figure 48 show excellent perspectives of blast cast results.

The data does show a general relationship between D/W and percentage cast for the various sections. Figure 49 summarizes the incremental drilling, explosives and accessive costs in generalized terms over a standard production shot. The incremental benefits to volumes cast are indicated also.

High speed film analysis of targets suspended on the face at the north end of the shot indicated face velocities of 17.5m/s, 25m/s, 28m/s, and 20m/s at 5, 10, 15, and 20m from the collar respectively.

The viewing perspective of the locam camera was such that collar movement and flyrock generated was not easily observed. However, observations of the video tape indicates that there was no significant flyrock or rifling in the blast cast area suggesting that the collars were not too short.

Figure 46 LJ #1 BLAST RESULTS

			2 D ANALYSIS				3 D ANALYSIS			
Section	D/W	Pre Blast Cast BSM	Cast Beyond M Seam BSM	% Cast	Cast Beyond L Seam BSM	% Cast	Cast Beyond M Seam BSM	% Cast	Cast Beyond L Seam BSM	% Cast
1	N/A	N/A	323	N/A	231	N/A				
2	0.94	1,069	600	56	455	43	819	77	568	55
3	0.78	1,000	611	61	475	48	774	71	559	56
4	0.73	1,048	518	49	392	37	518	49	392	37
5	0.63	808	483	60	322	40	483	60	322	40
6	0.68	517	260	50	166	32	260	50	166	32
7	N/A	N/A	69	N/A	58	N/A				

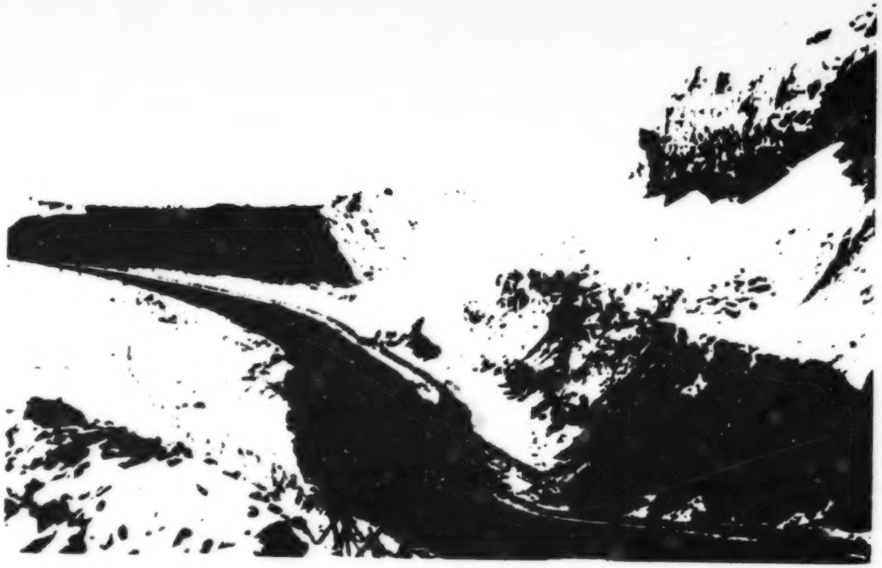
Total Volume, Vt = 105,085 BCM

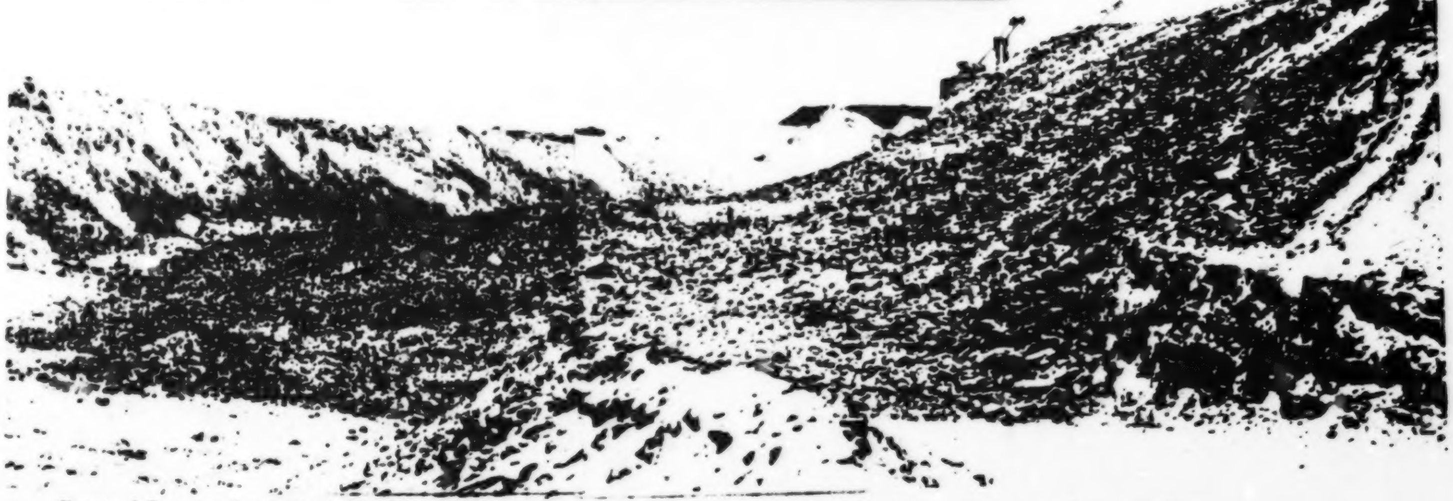
From 2D Analysis: Volume Cast, Vc_M, beyond M seam = 70,000 BCM
Percent Cast beyond M seam = 66

Volume Cast, Vc_L, beyond L seam = 51,200 BCM
Percent Cast beyond L seam = 48.5

Note:
- It is likely that a significant volume of waste from the adjacent production pattern was cast as well
- 3D analysis correction for fanning effect of material being cast off section. This provides a means of estimating percentage cast on a section by section basis.
- Section 7 was drawn through standard production pattern area not considered in incremental analysis.

Pre blast photograph.





Post blast photograph. Note the surveyor in the middle of the picture for scale.

Figure 47 L-J #1 Pre and Post Blast Photographs

Pre Blast

Post blast. Note surveyor on muckpile below and to the left of the truck.

Figure 48 L-J #1 Pre and Post Blast Photographs

Figure 49 L-J #1 Incremental Costs and Benefits

Incremental Costs

$35,055\ U_{2515} + 54,097\ U_{2510} - 58,231\ U_{2445}$ } Explosive cost

$+\ 48\ (U_p + U_n)$ } Accessory cost

$+\ 495\ U_D$ } Drilling cost

Incremental Benefits for L and M Seams respectively

$= (Vc_L - Vpc_L)U_{DLL} + (Vc_{ML} - Vpc_{ML})U_{DLM}$

$= (Vc_L - 0.7 * Vt * \%Vpc)U_{DLL} + (Vc_{ML} - 0.3 * Vt * \%Vpc)U_{DLM}$

- where 0.7, 0.3 are factors to proportion the benefit of the production cast volume relative to Vt between L and M seam, mined limits
- Vc_{ML} is the volume cast beyond M but not past L seam

$= (51,200 - 0.7 * 105,585 * \%Vpc)U_{DLL}$

$+\ (18,800 - 0.3 * 105,585 * \%Vpc)U_{DLM}$

For %Vpc = 0.15 estimated

$= 40,113\ U_{DLL} + 14,049\ U_{DLM}$

2.4 BLAST CAST SIMULATION RESULTS

With the assistance of CIL personnel, several simulation runs on the Sabrex, blast simulaton, computer package were made.

The simulations modelled the CC #2 and the L-J #1 blast cast. In the CC #2 simulation, actual face velocity information determined from high speed film analysis was input. This information was not timely for the L-J simulation.

Simulation results from CC #2 shot indicate 33% cast and the actual percentage cast was 31%. Complexities in the loading instructions and drilling details in L-J #1 pattern were such that the simulation model could not precisely match these conditions however an approximate model was developed. The results indicated lower face velocities and lower cast volumes than were actually achieved. This is probably a result of uncertainties in estimating effective burden when a portion of the material in front of the toe had been previously blasted.

Several test runs on Sabrex were done in order to examine the sensitivity of blast results with variations in hole inclination, explosive bulk strength and D/W ratios. One of the major surprises that resulted from this work was the relative insensitivity of percentage cast to different bulk strength explosives given the same pattern spacing. Higher bulk strength explosives did provide noticeably improved fragmentation according to the simulation but resulted in very minor increases in volume cast.

3.0 DRILLING

3.1 BACKGROUND

Although outside the formal scope of the project, the experience and knowledge gained in drilling during the blast cast program is of interest. Specifically, an increase in the use of the bit diameter for the Bucyrus Erie 45-R drills, and large diameter rotary angle holes which have not seen common use in Canada warrants documentation.

The Colorado visit in August, 1986 had illustrated the effective use of angle holes with large diameter rotary drills. The Trapper Mine was utilizing a Bucyrus Erie 45-R with 270mm diameter bits to drill angle holes and the Colowyo Coal was installing a conversion package on their 45-R drills to permit angle hole drilling (again with 270mm bits).

3.2 BUCYRUS ERIE 45-R BIT DIAMETER INCREASE

A study initiated following the Colorado field trip indicated that the bit diameter of the Fording Bucyrus Erie 45-R drills could be increased from 250 to 270mm. This was in part attributable to the large capacity (1,350 cfm) compressors installed on the drills at purchase. Adequate bailing velocity was available after derating for elevation even without a change in steel size. The modification was carried out on all three 45-R's by January 1987 following a test period on a single drill.

All drills have performed well with an overall increase in productivity[1] as represented in Figure 50. The improvement related to the bit size increase is thought to be a reflection of bit insert configuration, adequate pulldown weight given Fording's rock characteristics, and excess bailing capacity. Other factors such as operator performance and a reduction in redrills are also significant.

The bit size change increased borehole area 17% from 491 cm^2 to 573 cm^2. As a result, a conservative scaling factor equal to the ratio of the hole diameters to the two thirds power was applied to produce an equivalent powder factor in a new pattern size. The pattern size for 45-R drills was therefore increased from 8x8m to 8.5x8.5m, a 13% increase in yield. Drilling economics were similarly improved.

3.3 ANGLE HOLE DRILLING

The importance of angle hole drilling was recognized at the outset of the project both as a means of reducing toe burdens and introducing a vertical component into blast trajectories. This was supported in the literature search and in discussions with consultants. Since Fording lacked the ability to drill angle holes with its fleet of rotary drills comprised of three Bucyrus Erie 45-R drills and four Bucyrus Erie 60-R drills, it was decided to subcontract the angle drilling.

A cost plus contract for angle holes utilizing a down-the-hole (DTH), truck mounted Schramm drill with 270mm diameter bits and 102mm diameter steel was developed. Initial problems on Eagle Mountain with hole collaring and

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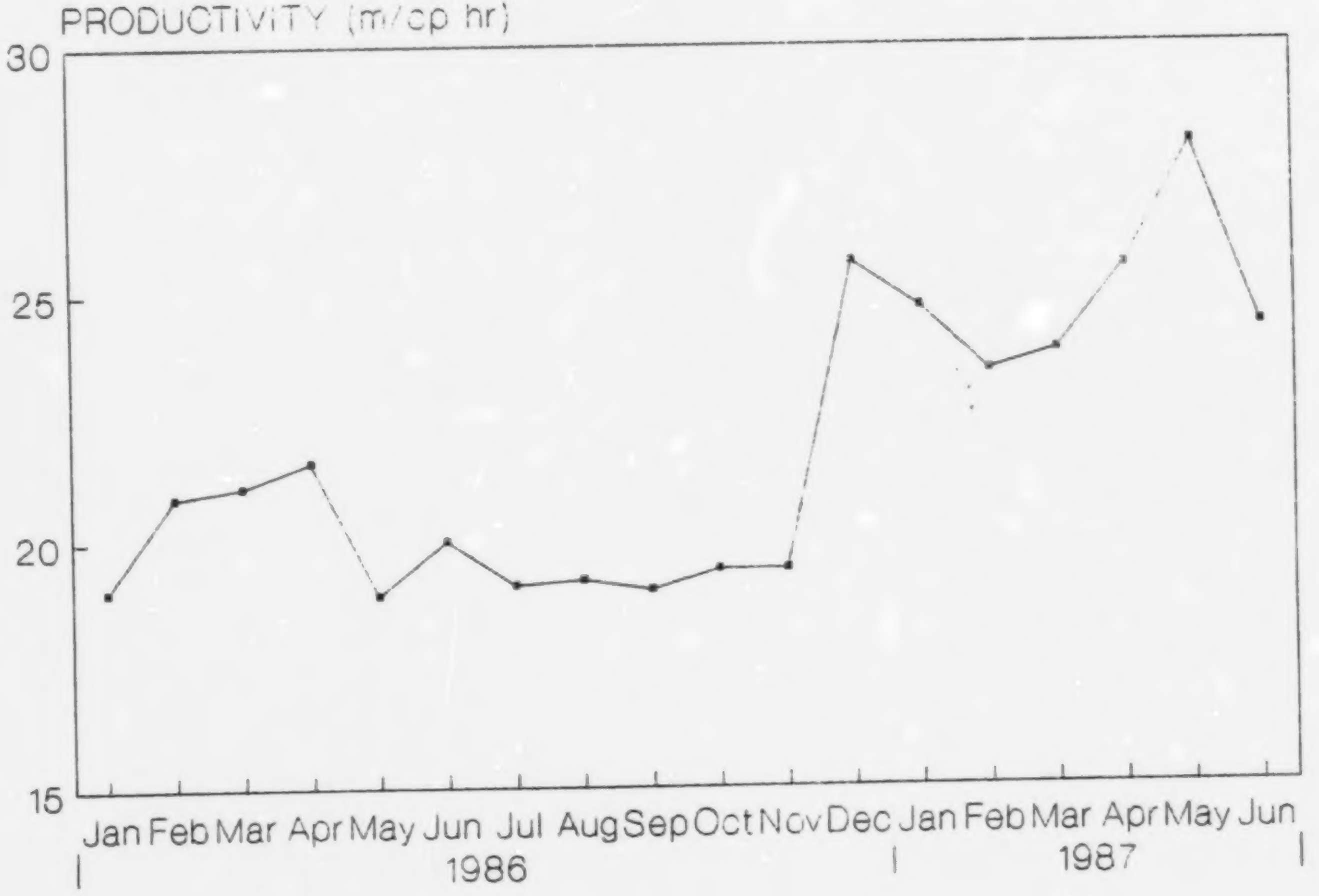
[1] productivity in net meters per operating hour = total meters - (redrills + coal + overdrilling).

Figure 50

45-R DRILL PRODUCTIVITY



drilling were attributable to inadequate bailing velocity. This pattern was abandoned after several days of unsuccessful drilling and the rig moved to Brownie Pit.

The drill was set up on BR-Y #1 pattern with an auxiliary high capacity compressor to increase bailing velocity. Although the portable compressor provided adequate theoretical bailing velocity, problems with collaring and caving holes were still encountered. The DTH hammer also created problems, further contributing to the difficulty with collaring holes. This operating problem may have been related specifically to geological bedding structure, and to a certain extent, the large subgrade of 2.5m utilized on previous benches as per Fording's standard practice. Difficulty in collaring or maintaining the integrity of drill holes is not a common problem with Fording's rotary drills. Although the BR-Y #1 pattern, and subsequently BR-A #2 were successfully completed, the overall productivity and resulting cost were deemed unacceptable.

The hiatus in the testing program from November 1986 to September 1987 afforded the project team to opportunity to consider other angle drilling alternatives. Relying again on the Colorado Field trip, consideration was given to modifying one of the Bucyrus Erie 45-R drills on site. In June 1987, a justification was developed, with a decision and issuance of a purchase order following in July 1987. The payback was estimated at approximately one year (very sensitive to usage) based on an initial cost of $60,000 including parts, labour, and training.

Delays in the delivery of parts until early November held up the conversion. The actual installation on #2 Drill (94502) was completed in approximately one week. The drill was returned to service on November 13, 1987.

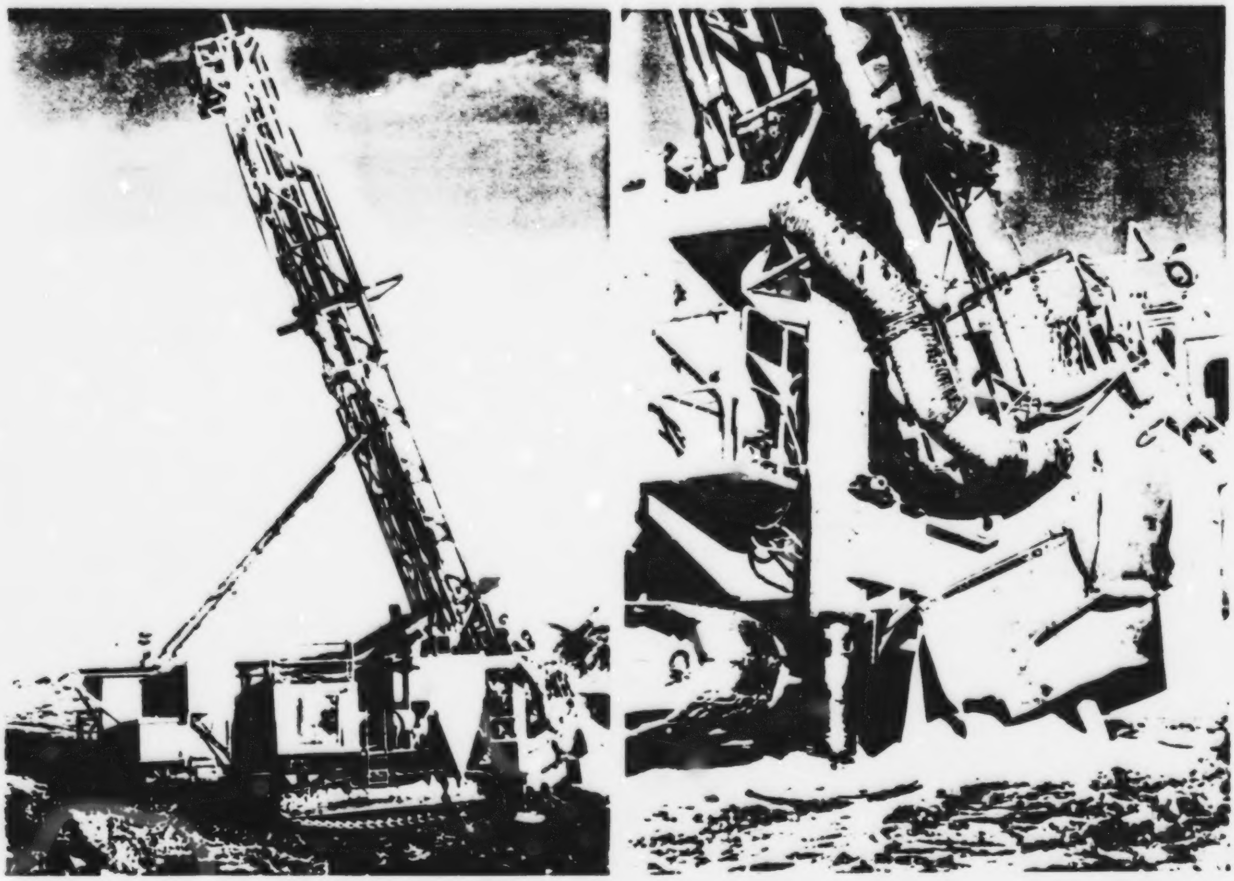
The angle hole conversion consists of a change to two main areas of the drill. The first is a modification to the drill steel rack assembly. Parts include a pipe positioner, air valves, rotochambers, operating levers, and position arm which allow steel changes with the mast in an inclined position. The second modification is in the mast support with the addition of two mast braces tied into side frames which can be set from 0 to 30° from vertical in 5° increments. Changeover in the field involves adjusting the mast angle with conventional tilt cylinders and locking in the braces with pins. Total conversion time from vertical to angle hole drilling requires roughly one hour. A photograph of the drill set up on angle hole is shown in Figure 51.

Performance of the converted drill has been very good to date. Two patterns, C-C #2 and L-J #1, drilled in November and early December have included angle holes. Although hole collaring and caving were a concern, relatively few problems were encountered. The initial pattern, C-C #2 even included several meters of rehandle at the collars of the hole which did not significantly affect the drill performance.

Productivity on C-C #2 and L-J #1 averaged 25.5 meters per operating hour (m/op.hr.) and 27.5 m/op.hr. respectively for combined vertical and angle holes (10, 15, and 20°). This compares with a year to date average of 26.0 m/op.hr. prior to the angle hole conversion. Although less than 20% of the total holes in these two test patterns were angled, and several other variables (ie. training, ground conditions, etc.) affected drill performance, it appears that angle hole drilling will have adverse affect on productivity, apart from the decreased penetration rate.

It is anticipated that operating costs will rise slightly with angle hole drilling due to bit shirttail and steel/stabilizer wear, and the increased strain on the mast assembly. This has not been quantified to date.

Overall, the modification of #2 Drill to angle hole capability appears to have been successful. Angle hole blast casting has proven effective and will see increasing use in the future as a standard practice. Other uses of angle hole drilling which have not yet been tested include drilling on topographic slopes where drill ramp requirements can be reduced, and drilling footwall wedges of steeply dipping seams once the coal has been removed (thus reducing mining dilution).



Left picture shows profile of 2 drill nosed out to the crest drilling 20° holes. Note the arms to support the mast adjustable to 5° angle increments.

Right picture shows the front end of the drill. Note the extended dust curtain and the support arm used to guide the steel when steels are being changed.

Figure 51. Angle Drilling on Eagle Mountain

4.0 CONCLUSIONS

4.1 ECONOMIC ANALYSIS

Net Savings/BCM

A general expression for incremental costs and incremental benefits was developed for each blast cast in the pertinent blast report. Given a set of economic and physical parameters, the net savings per BCM of blast cast waste was evaluated according to the formulae:

$$\text{Net Savings/BCM} = \frac{\text{Incremental Benefit} - \text{Incremental Cost}}{Vt}$$

This formulation is consistent with equation 2.

The necessary economic parameters include unit drilling costs, unit explosive costs, and unit equipment operating costs for a shovel or a dragline according to what is applicable.

Physical parameters include a swell factor for the open type blast casts (BRA #2, BRA #3, CC #2) and an estimation of the percentage cast on a production shot with incremental costs equal to zero by definition.

The individual economic parameters are confidential and therefore only approximate values have been used in the analysis. The physical parameters assumed in the analysis were set as indicated in Figure 52.

The Net Savings per BCM for each blast cast is listed in Figure 52. L-J #1 and L-G #2 were the only two blasts for which a net savings resulted with L-J #1 having the largest net unit savings of 0.106 $/BCM. The remaining blast casts were unsuccessful according to the economic models used. CC #2 and L-G #4a were fractionally unfavorable. BRA #3 and L-G #1 were the most unfavorable with a net unit cost in excess of 0.095 $/BCM.

Figure 52 Summary of Blast Cast Results

Blast	D/W[1]	$\frac{L-L_s}{L}$[1]	EF WEU/BCM	Angle Drilling Y/N	% Dry[2] Hole Product %	Front Row Toe Burden Details		Percent Cast		Approx.[3] Net Savings $/BCM	Most Sensitive Variables[4] (1) (2)	Likelihood[5] Benfit Could Change Sign
						Ave. Toe Burden (m)	Comments	Blast Cast	Equivalent Production Shot (Est)			
BRY	0.3-0.43	0.55	58	Y	98	3	13m additional broken rock	N/A	N/A	-ve	N/A	Nil
BRA #2	0.38	0.63	91	Y	100	11.2	unknown % of this OB	19	5	-0.059	Explosives Cost, Swell Factor	Nil
BRA #3	1.1	0.65	89	N	100	17	unknown % of this OB	34	7	-0.111	Explosives Cost, Swell Factor	Nil
CC #2	0.6	0.71	86	Y	10.7	17	Variable from 10m of solid rock to 20m with unknown amount of OB as a portion of this.	31	7	-0.013	Swell Factor, Expl. Cost, Op. Cost	Moderate
L-G #1	0.53	0.63	90	N	27	11	An additional 7-25m of blast rock and OB against toe	12	5	-0.095	Expl. Cost, Op. Cost	Nil
L-G #2	0.63	0.65	89	N	88	9m	An additional 8-10m of OB against toe	50	10	+0.025	Op. Cost, Expl. Cost	Low
L-G #4a	0.68	0.65	82	N	0	13m	An additional 7-10m of OB against toe	29	7	-0.020	Expl. Cost, Op. Cost	Low
L-G #4b	0.77	0.71	86	N	0	16	An additional 7m of OB against toe	25	7	-0.058	Expl. Cost, Op. Cost	Nil
L-J #1	0.76	0.79	76.5	Y	100	8m	An additional 10m of blasted rock and OB against the toe	66	15	+0.106	Op. Cost, Expl. Cost	Nil

- Physical Parameters, S = 1.2, % Vpc as indicated for each blast in the above table

[1] Fraction of Hole length loaded with explosive
[2] Based on weight not percentage of holes
[3] Based on the assumed input parameters. FCL unit costs are confidential therefore exact costs are not used.
[4] For variables analyzed in sensitivity analysis: Unit equipment cost, drilling cost, explosive costs, % Cast in corresponding production shot, swell factor
[5] Based on no design changes and with a qualitative assessment of the sensitivity analyses

It is worth emphasizing that the benefits derived for casting waste assume either dragline costs or shovel side casting costs. If the valid comparison is a truck/shovel operation with a long haul the benefits for casting each BCM (U_{TS}) could be much larger than the values used thereby increasing the net value of each blast cast.

Sensitivity Analysis

Sensitivity analyses of net savings per BCM to variation in values of selected economic and physical parameters were carried out. The variables analyzed were:

swell factor }
Percentage Cast for Standard Production Shot (%Vpc * 100) } physical

Drilling Costs }
Explosive Costs } economic
Equipment Operating Costs }

The range over which each variable was tested is related to the accuracy with which each parameter is known. Of all the variables, %Vpc is estimated with least confidence. The range of variation selected was ±50%. In reality this parameter could easily vary by much more than this.

Swell factor was varied by ±10% and all the remaining variables were varied by ±20%. Drilling costs and explosive costs are known accurately however approximate values were used in the analysis. The wide range in variance tested incorporates actual costs and is useful in order to extrapolate results to other pit areas where costs may vary significantly. Average unit operating costs are known accurately but the effects of higher energy factors on unit operating costs would likely cause a reduction in these costs. The benefits of reduced operating costs as a result of better digging conditions are not built into the analysis. The analysis assumes digging costs in a production shot and a blast cast shot are identical. The variation in operating costs analyzed in the sensitivity analysis represent the base value. As these costs rise, the net savings of the blasts increase because the value of each BCM cast is higher.

Figures 53 through 60 are the results of the sensitivity analysis. As indicated in Figure 52, operating cost and explosive costs are in general the most sensitive to the Net Savings per BCM. For the open cast blasts Swell Factor is also a sensitive variable.

After a qualitative examination of the results of the sensitivity analysis it is apparent that CC #2 and L-G #4a would have a moderate to low likelihood of becoming favorable with reasonable variation in parameter values. The remaining unfavorable blast casts are clearly unfavorable.

The net savings/BCM appears to be insensitive to the %Vpc for the range analyzed. However, the likelihood of %Vpc exceeding the ±50% range is considered large in view of the qualitative estimates made. This implies that %Vpc is probably more sensitive than it appears especially if probability of each level of variation for each of the variables could be taken into account.

4.2 GENERAL RESULTS

The data tabulated in Figure 52 indicate several points of interest which must be noted. This data was plotted as % Cast versus D/W in Figure 61. There is a positive correlation between D/W and the percent cast. Several variables tend to cloud this relationship. Note that all blast casts except BRY have approximately the same energy factors with a mean of 86 eu/BCM.

The results indicate that casting suffers when large quantities of Magnafrac type explosives are used as compared to Heavy ANFO explosive types. This is consistent with the HE/SE partitioning between Magnafrac and SuperAN explosive types. CC #2, L-G #4a, and L-G #4b were loaded with 100% Magnafrac. Furthermore, the cost of achieving a percentage increase

Figure 53

BR-A #2 SENSITIVITY ANALYSIS

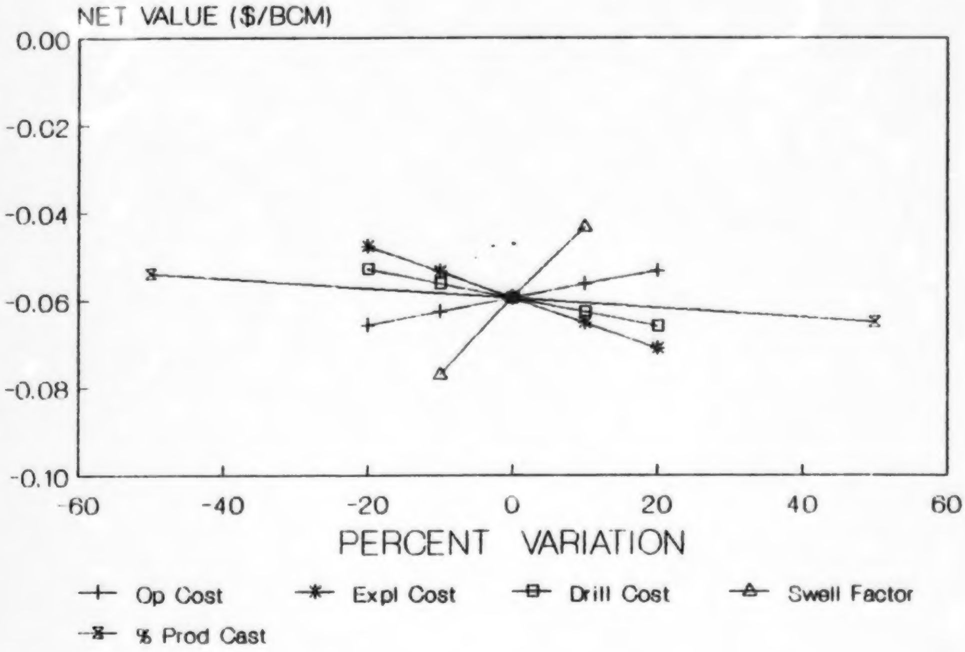


Figure 54

BR-A #3 SENSITIVITY ANALYSIS

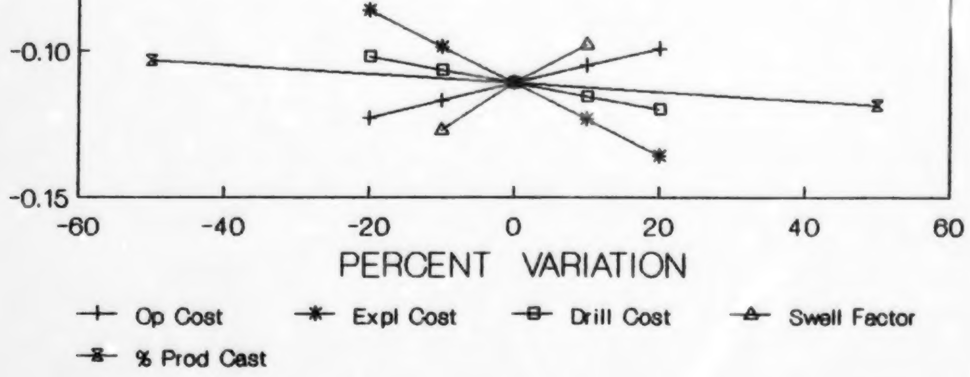


Figure 55

CC #2 SENSITIVITY ANALYSIS

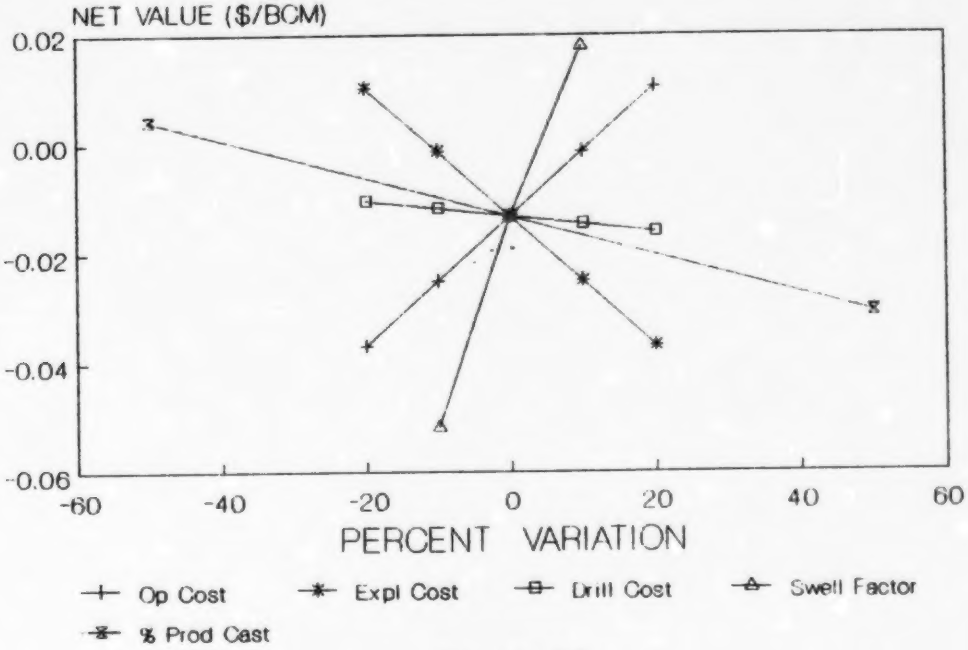


Figure 56

LG #1 SENSITIVITY ANALYSIS

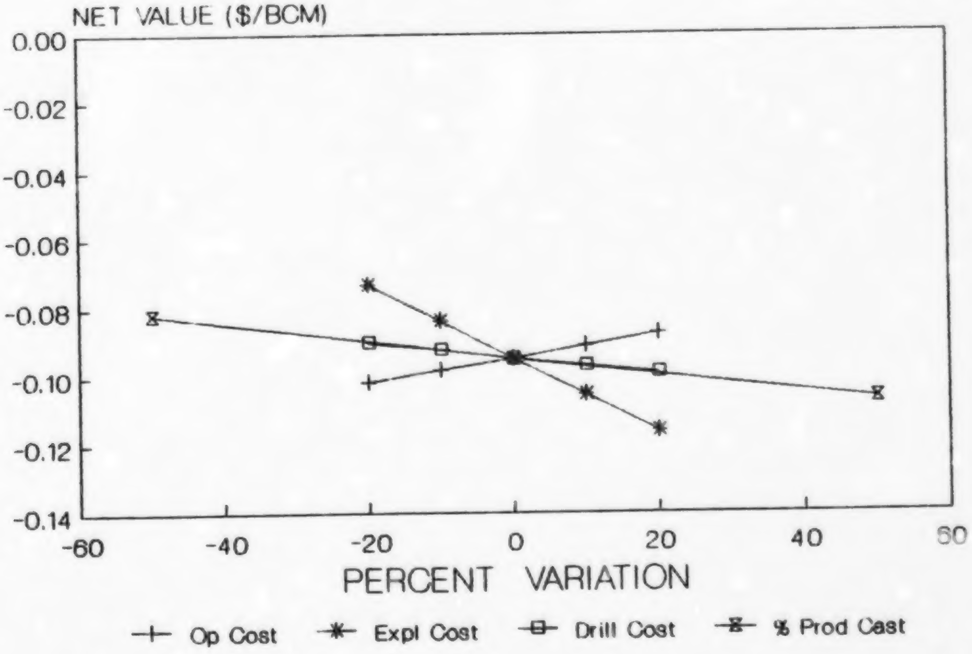


Figure 57

LG #2 SENSITIVITY ANALYSIS

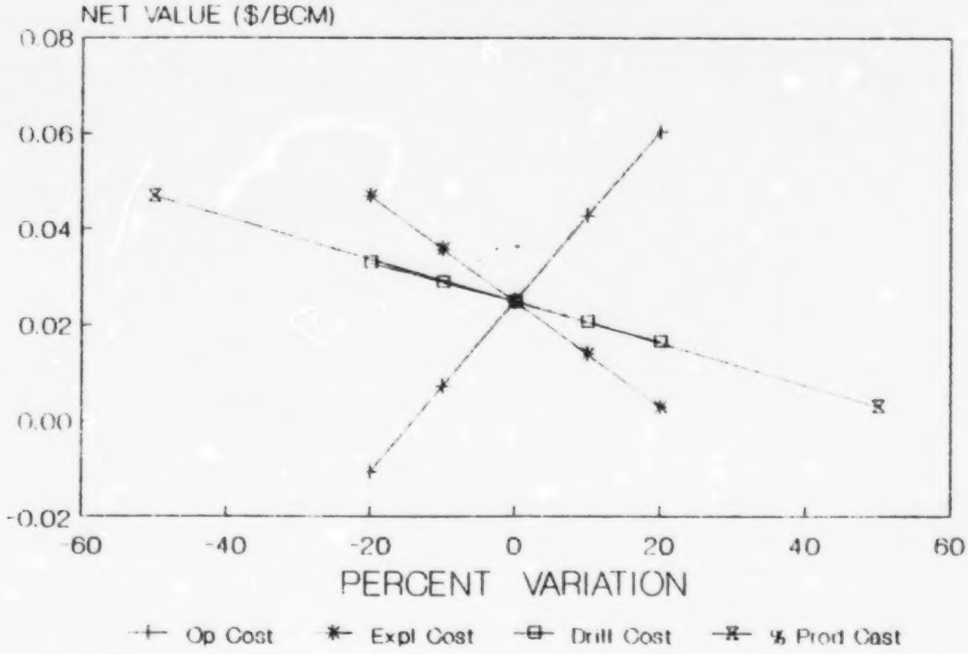


Figure 58

LG #4A SENSITIVITY ANALYSIS

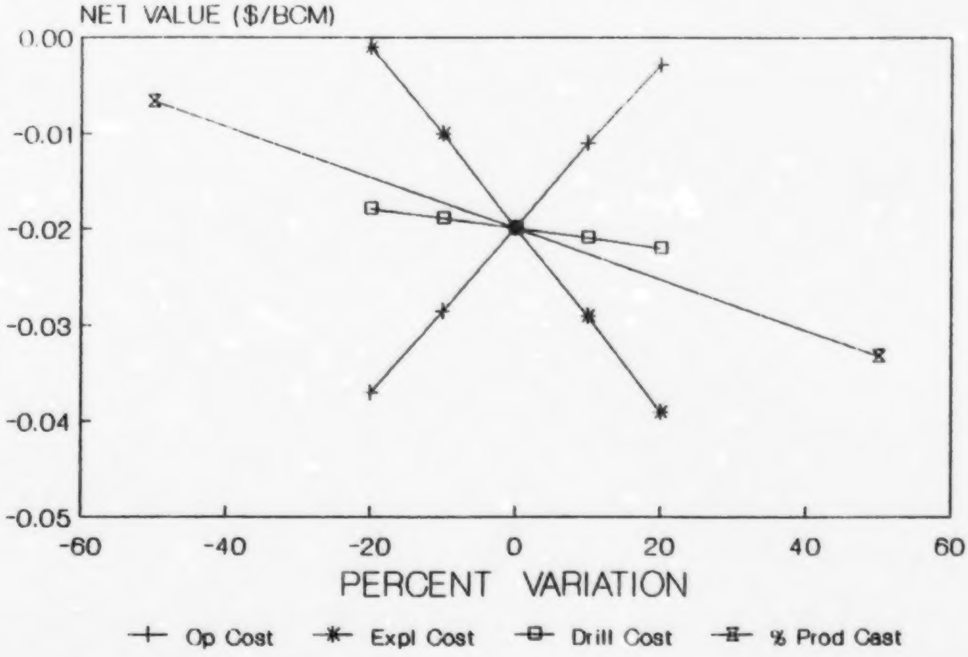


Figure 59

LG #4B SENSITIVITY ANALYSIS

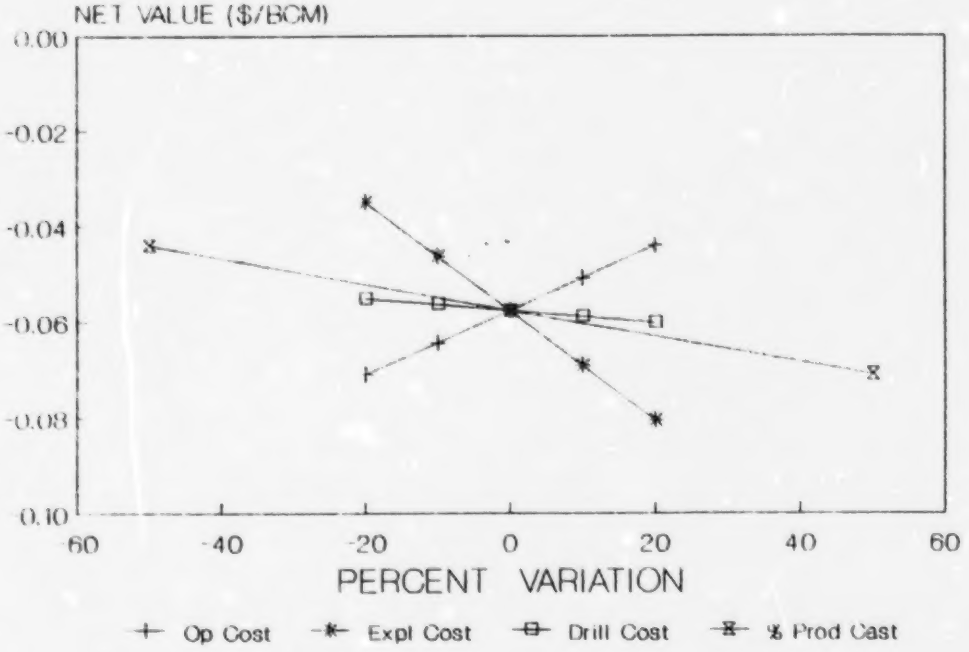


Figure 60

LJ #1 SENSITIVITY ANALYSIS

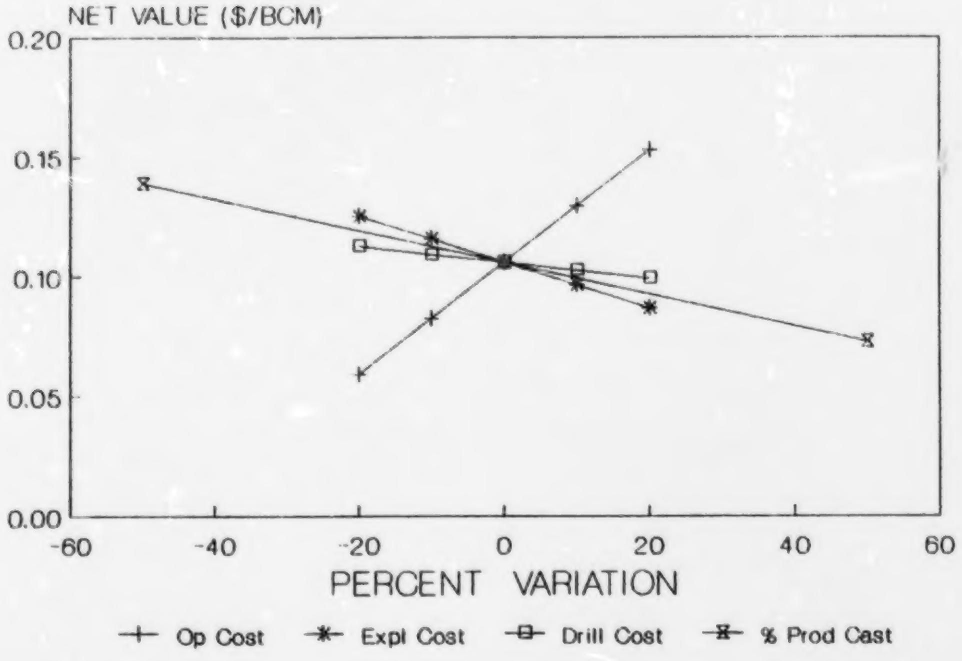
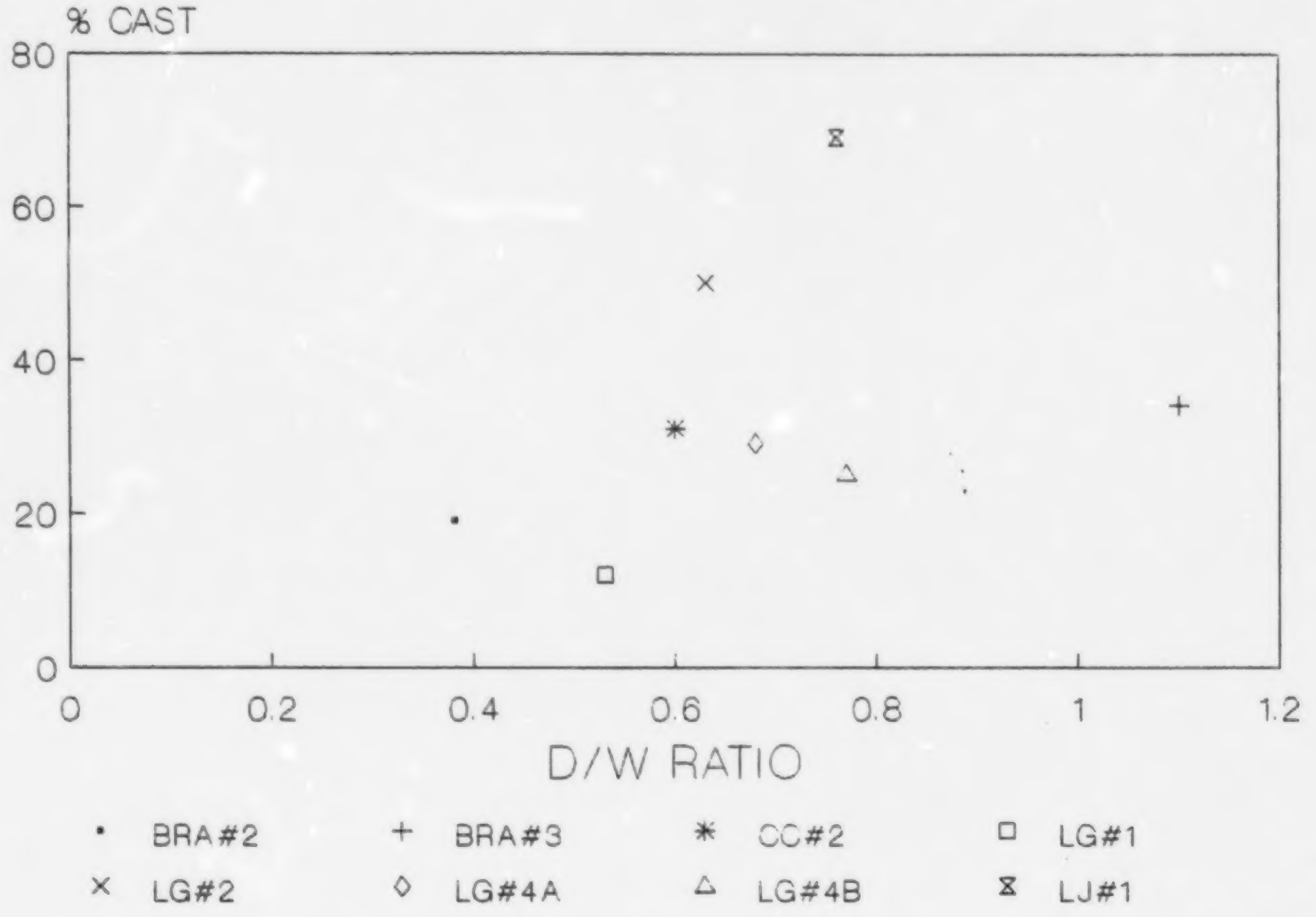


Figure 61

%CAST versus D/W RATIO



in bulk strength with wet hole explosive types may be more expensive than with dry hole explosive types.

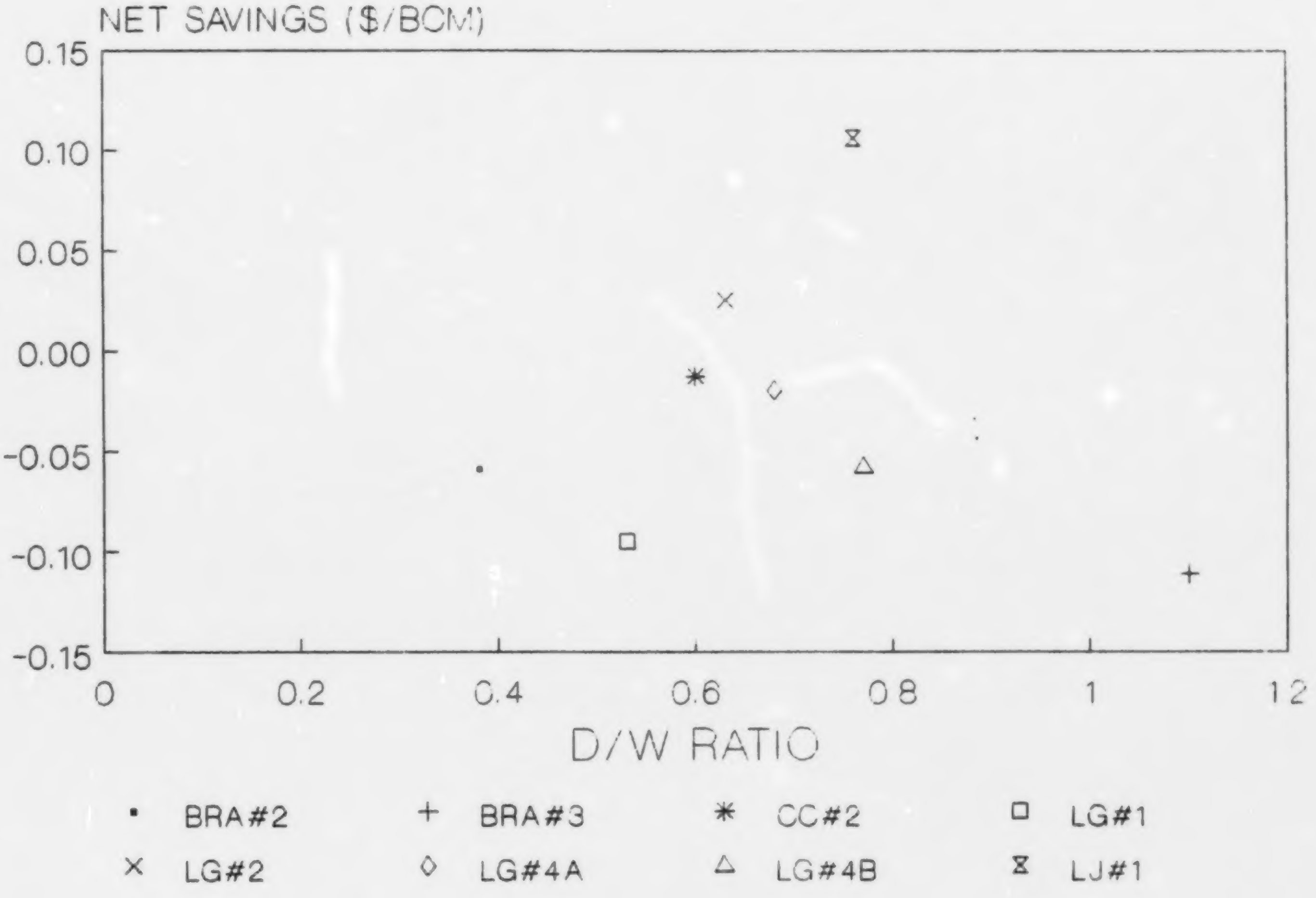
Blast cast results appear to be extremely sensitive to the toe burden on the front row of holes. All cast blasts with unfavorable net savings/BCM had excessive toe burdens. The most unfavorable shots had the largest toe burdens. The importance of reasonable toe burdens cannot be understated. Results plotted as Net Savings/BCM versus D/W, plotted in Figure 62, confirm the importance of explosive type and toe burdens. Net Savings/BCM are positive for D/W larger than approximately 0.6 when burdens are reasonable and holes can be loaded with Heavy ANFO. These results also indicate that where burdens are excessive and/or holes are loaded with Magnafrac, casting is not beneficial for D/W ratios in excess of 1.0.

All of the above information is invaluable to future casting efforts at Fording River Operations. It is also important to recognize the measures which can be instituted to enhance casting that result in no incremental costs. These measures should obviously be instituted in all situations. They include blast direction perpendicular to the free face, increased delay periods to 100ms for rows adjacent to the crest and angle drilling if the unit cost of angle drilling and vertical hole drilling is roughly equivalent.

In many situations where an open free face exists and toe burdens are equivalent to normal toe burdens, larger benefits will likely be realized with small incremental cost measures such as raising collars to the minimum consistent with containment and perhaps heavier toe loads. For D/W ratios in excess of 1.0 and for dry holes the probability of achieving favorable economics is large especially for 45R pattern sizes in these situations. The above comments must be tempered with site specific, pit specific, and area specific benefits for casting waste volumes.

Figure 62

NET SAVINGS ($/BCM) VS D/W RATIO



4.3 BLAST CAST SIMULATIONS

The Sabrex computer simulation package proved to be an excellent tool for blast cast simulation. Results in general compared closely with field results. Significant deviations occurred, as expected, when field conditions could not be properly modelled. Actual drilling and loading details in some instances were too complex to model. Furthermore the actual effective toe burdens were uncertain in many instances when unknown thicknesses of overburden and or quantities of blasted waste were left against the toe. Three dimensional affects of rock knobs and L/W ratios are probably not accounted for in the model. Additionally representative rock strength characteristics to be input into the model were estimated without the benefit of laboratory tests.

Sabrex could have been used more effectively to provide more feed forward information to blast design as well as provide feedback information for comparison studies.

4.4 DRILLING

Blast Casting requires more drilling resources than production blasting. If significant blast casting is foreseen, a careful study of available drilling resources is necessary. A successful casting program will demand more drilling resources and release waste moving resources.

At Fording River Operations drilling productivities have risen sufficiently to allow limited additional drill resource usage on blast cast patterns.

Angle hole drilling in blast cast situations is one of the most important design characteristics to maximize savings. With the exception of severely caving ground conditions the cost of angle drilling and vertical hole drilling is expected to be roughly equivalent. The enhancement afforded to casting results with the use of angle holes is large. Angle hole capabilities enable the optimum utilization of a blast cast site especially where deep holes can be drilled. For example, without angle hole potential, casting over a high face may have to be done in two benches in order that front rows have a reasonable toe burden. In each bench Vt will be restricted by D/W ratios. Alternatively with angle hole capabilities the high faces, so important to successful casting economics, can be used. Vt will be larger for the constraining D/W ratio resulting in larger cast volumes at a given net savings per BCM.

Experience gained during this program indicates that different types of angle drills can have dramatically different success rates.

4.5 SUMMARY

A valuable method has been developed for assessing blast casts in terms of net value. Net value is the difference between incremental costs and benefits. Incremental costs are attributable to higher powder and energy factors. The incremental benefit is derived from waste volumes effectively cast relative to a conventional production blast and the applicable waste removal cost for equipment. The benefit is rationalized with respect to existing mine plans.

Only two test blasts resulted in a calculated net savings. This is a reflection that the work has not fully addressed the difficult problem of optimization, but rather has provided a direction for subsequent work.

The analysis concludes that the net value is particularly sensitive to assumed swell factor and the estimate of percentage cast for conventional production blasts. Blast cast effectiveness is also very dependant on physical geometry such as depth to width (D/W) ratio and the toe burden on the first row.

Although a positive correlation between depth to width ration and percent cast was established, no qualitative relationship was evident on the basis of limited testing. As a general guideline, net savings are positive for D/W ratio greater than 0.6 when dry explosive products (ie. ANFO, Heavy

ANFO) are used. Very large D/W ratios would be required to offset the costs of wet explosive products (ie. Magnafrac).

Several measures can be implemented to optimize casting without incurring incremental costs. These include blast direction perpendicular to the free face, row delays of 100ms, and angle hole drilling.

Although the problem of blast cast optimization has not been fully addressed, the guidelines have been established. Given a reasonable set of geometric conditions, blast casting can be effectively applied to move material at the lowest overall cost. The economic potential for blast casting in Canadian surface coal mines has been demonstrated.

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